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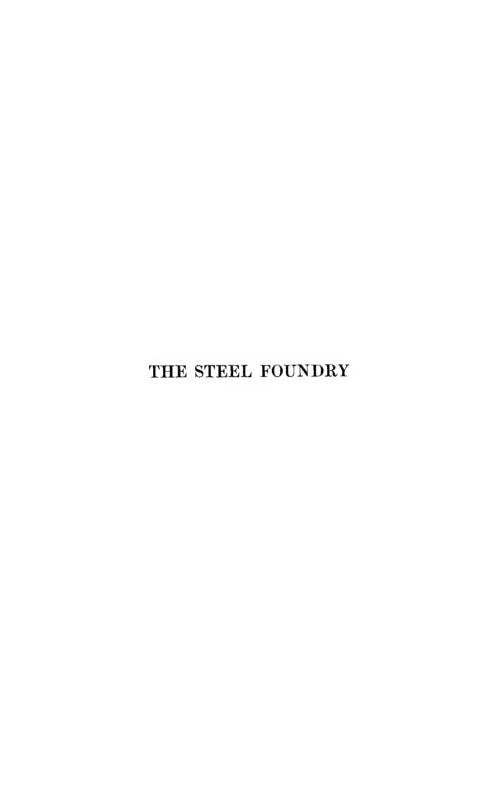
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THE STEEL FOUNDRY

BY JOHN HOWE HALL

METALLURGICAL ENGINEER, TAYLOR-WHARTON IRON AND STEEL COMPANY

SECOND EDITION

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To

HENRY MARION HOWE, LL. D.

UNDER WHOSE INSPIRING GUIDANCE IT HAS BEEN
MY GOOD FORTUNE TO WORK—THIS BOOK
IS AFFECTIONATELY DEDICATED

PREFACE TO SECOND EDITION

The tremendous increase in the number of electric furnaces in steel foundries since the first edition of this book was written in 1913, and the refinements in electric furnace practice that have resulted, made a revision of this volume advisable. In these nine years, too, much has been added to our knowledge of moulding sands, heat treatment and a great many other phases of foundry

ERRATUM

Captions for figures 43 and 44 have been transposed in error.

HALL'S-THE STEEL FOUNDRY

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J. H. HALL.

HIGH BRIDGE, N. J. March, 1922.

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The work of revision has, of necessity, been done in what little spare time it has been possible to find, and it is the hope of the author that the result will justify the effort expended.

Thanks are due to the late R. L. Lindstrom, whose untimely death is a loss to the foundry industry, for permission to reprint the photographs and analyses of foundry sands; to the American Bridge Company, the American Institute of Mining and Metallurgical Engineers and the American Foundrymen's Association for the loan of cuts and permission to reprint extracts from some of the author's papers; to Carl Koch and Arthur H. Jameson for valuable advice and suggestions; to the Taylor-Wharton Iron & Steel Co. for permission to publish certain data referring to manganese steel; and to the author's associates in that company, especially G. R. Hanks, R. D. Jordan and Wallace G. Berlin for suggestions on many points.

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PREFACE TO FIRST EDITION

In this work the object has been to set forth the metallurgy of the steel foundry from the point of view of the engineer who keeps constantly in mind that his is a profession whose usefulness consists in prescribing the cheapest means of producing objects or structures of sufficient excellence for the purposes for which they are intended. The aim of the engineer in his work should be three-fold—to produce an article that will completely serve its purpose, to provide in the object the highest excellence that is justified by the use for which it is intended (and hence the price that can be paid for it), and to produce these objects at the least possible expense.

In the present volume are considered the classes of steel castings that are in demand in this country to-day, and their characteristics from a manufacturing point of view; the types of steel-making processes that are in use, and their characteristic features, such as relative cost of installation, relative cost and quality of steel produced, etc., that prescribe the use of one or another for manufacturing the sort of castings desired; and the procedure throughout the shop, such as moulding, pouring, annealing, etc., in the light of its influence on the quality and cost of the product. It is the hope of the author that this volume will prove of interest and benefit to steel foundry metallurgists and superintendents, and to the managers. officers and stockholders of existing or projected foundries; and to their judgment he submits these pages, hoping to be repaid for the labor of authorship by the knowledge that they have assisted to some degree the efforts of his co-workers in the great field in which we all labor.

The author wishes especially to thank Professor Joseph W. Richards, Professor Albert Sauveur, Professor Henry M. Howe, and Mr. Arthur Simonson for valuable advice and counsel; and Mr. A. H. Jameson and the David Williams Company for permission to reprint the article from the "Iron Age" on the building-up of impurities in steel. Certain of the micrographs and data in the chapter on heat-treatment have already been published in the author's papers before the American Institute of Mining Engineers and the American Society for Testing Materials; they are republished here with more detailed comment and explanation.

New York City, January, 1914.

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THE STEEL FOUNDRY

CHAPTER I

INTRODUCTORY

The function of the steel foundry is to produce by casting in sand or other moulds, steel shapes that cannot be rolled or forged; or that would require special equipment for rolling and forging, that is not justified by the number of the particular article wanted. Thus many steel articles that would be made by drop forging or press forging in dies, if wanted by the hundred or thousand, are made of cast steel when not enough are to be purchased to pay for the expensive dies required. The foundry, therefore, supplies the country with steel shapes often more costly than rolled or forged shapes; and herein lies one of the most vexatious features of the business.

In general, steel foundries may be roughly divided into three classes, according to the market in which they sell their castings. These are, first the Tonnage Foundry, that is the foundry supplying great numbers of castings from relatively few patterns, generally of shapes that can be made only of cast metal, and frequently bidding for long term orders on a flat rate per pound of steel. Such are the foundries furnishing draw bars and knuckles, bolsters, etc., for railroad cars, frames, driving wheel centers and other castings for locomotives, and other work of that general nature. The problem of these foundries in working to make a profit is similar to that of the rolling mills, and may be summarized as that of turning out a heavy enough tonnage of comparatively simple work at a small profit per pound, to return a dividend on the investment. The effort in these shops is concentrated on producing the greatest possible number of castings per day, and everything must be arranged to keep a constant flow of work passing through every department—delays are fatal. ways the problems in these shops are the easiest of solution to be met in the business, since the constant production of one class of castings lends itself naturally to the introduction of labor-saving devices suited to the particular sort of work being handled, and from the

familiarity attained by constant repetition the men throughout the shop learn naturally the quickest ways of performing each detail of the operations. Raw materials, tools, equipment of every sort can be standardized, processes adopted or rejected according to how well they fit in with the primary object of keeping the output moving steadily through the shop, and the operation of the plant made almost automatic. The desired quality of the product is known, and improvements consist almost entirely in devising means of producing a superior article without interfering with the routine (and hence raising cost), in order that the salesmen may have an additional argument in favor of their goods and not be forced to shade prices to the last inch to get orders.

The second class is the Jobbing Foundry, a familiar name and a most expressive one, as shops of this class handle all the odd jobs that come along. Here are brought the miscellaneous machinery castings, automobile and truck castings, shapes that would be made by forging or rolling if wanted in quantity, and job lots of all sorts. The jobbing foundryman has a very different problem on his hands from that of the tonnage man, and his position is often an unenviable one. Frequently, one might almost say invariably, the number of castings to be furnished from each pattern is so small that a very few spoiled castings spell loss instead of profit on the order. The tonnage manufacturer has the opportunity to experiment with his patterns, change sections here and there, add fillets and brackets where needed, and generally get posted up by preliminary experiments on each pattern, and then start in and turn out finished work almost without thinking about details. The jobbing expert, on the other hand, must constantly foresee the difficulties to be encountered by the light that is in him, and must know almost by instinct the precautions to be taken in moulding and pouring to get a solid casting, the amount of metal that will be required for sink heads, gates, etc., and thus be able to fix a price that will yield him a profit. This price generally has to be so fixed as to cover the inevitable lost castings, which no human foresight or experience can entirely prevent; and the jobbing foundryman above all must have an unerring eye for the casting that from some peculiarity of design cannot be cast at all without a large number of "wasters," and must persuade the customer to alter the fatal features of the design, or turn the order down altogether. These are the cats and dogs of the business, and many are the wiles of the casting buyer in foisting them upon the unwary at prices that mean loss.

Another woe of the jobbing foundryman is the small casting that really should be a forging, on which he expends money for patterns, or for wasters made in experimenting on a small order, counting on subsequent orders produced without trouble to reimburse him—only to find that the machine of which his castings form a part is a success and is to be manufactured in quantity, justifying expense for dies to forge the parts and leaving the foundryman a sadder and a wiser man. The jobbing man's one really happy time is when business is so good that everyone is rushed to the limit, and orders are placed chiefly on quick delivery; then the overflow from the tonnage foundries comes to his shop and he waxes fat on standard work that can be easily and cheaply produced, and is paid his high prices because the work is wanted at once.

Our third class is composed of the foundries making a specialty of some alloy steel of particular excellence for certain purposes, or of some class of difficult castings especially suited to a particular steel-making process, and hard to make by any other. Almost as a matter of course the output of these foundries is a small one, and the profit per pound large. If the output were large, the tonnage foundries would gobble the work, especially when the specialty is in the shape of the casting rather than in the kind of steel; a large output in sight for a few difficult shapes of course justifying the installation of the special method of making steel needed, and the production of the work in quantity at a small profit per pound.

The Specialty Foundry, therefore, may partake of the characteristics of the jobbing foundry or of the tonnage foundry, according to the nature of the specialty—steel, or shape of casting. The foregoing remarks upon the two classes apply with equal force to this third class, and in the case of special steel the remarks upon the difficulties of jobbing work apply with even greater force, since to the difficulties of producing constantly varying shapes of work are added those of handling a steel frequently troublesome, for one reason or another, to produce even in simple shapes. Many an unwary foundryman has learned to his sorrow the difficulty of getting a share of the business of the manufacturer of special steels, who sells his product at a high figure because he must do so to make a profit, and is protected from competititon by the smallness of his tonnage and the knowledge of the fine points of his process gained from years of experience. When business is very dull, and the competition for orders very keen, so that price (and hence quality), are hammered down to the last possible inch, the payment of high prices for the output of a particular foundry means that that foundry has mastered a specialty that cannot be made on a "tonnage" basis, and competitors feeling around for business to fill up their shops in slack times should be very sure of their ground before going after a share of the specialty man's business, lest they get caught in the unforeseen difficulties of the specialty, and suffer considerable loss before they quite know how it happened.

Naturally, no hard and fast lines can be drawn between the three classes of foundry work into which the field is divided, and a particular shop may do business more or less along two or three of the general lines. Many a jobbing foundry has its specialties, which help to pay the profits of the business; many specialty foundries do a jobbing business to increase production and cut down overhead expenses; and either one may have standard lines that can be and are handled by methods similar to those of the tonnage shop. The management of a projected foundry, or of a foundry which is not paying the profits it should and is being reorganized, will do well to scrutinize their market closely and ascertain definitely the class in which they belong, in order to determine the direction in which their efforts should be directed to develop the business to the best advantage. For instance, the producers of a special steel, of which they have long enjoyed a monopoly, may find competition springing up and cutting seriously into their volume of sales. Investigation will probably show that they are in the habit of accepting orders only for such work as can be easily turned out; that methods of handling raw material and keeping product moving through the shop are archaic; and possibly that cheaper methods at various points of the process can be adopted to the betterment as well as cheapening of the product. The shop has been a pure specialty maker, and has neglected much of the jobbing work that it might have handled at a profit, and has not adopted the more modern methods that enable tonnage work to be produced cheaply. By improved handling methods, much of the output can be made at considerably reduced costs, and by accepting a wider range of orders for miscellaneous work which can be obtained at high figures, output will be increased and the cost of steel correspondingly decreased.

New companies should be particularly cautious in entering the field without sure knowledge of the conditions to be faced. The tonnage shop, of course, is seldom started up except by large interests backed by ample capital, and often closely allied with consuming industries, so that the organizers have an assured tonnage in sight

and are perfectly cognizant of the nature of their problem. Not so the small companies organized to carry on a jobbing or specialty business, often attracted by the supposedly fat profits of established companies, and incautiously entering the field with only sufficient capital to build a plant and carry on operations for a few months. The profits counted upon to carry on the business are in too many cases not forthcoming, due to losses arising from inexperience in fixing prices and handling odd castings in a jobbing business, from the unexpected difficulties encountered in the manufacture of special steels in complicated shapes, or what not. The troubles of many of the small foundries in the first few years of their life, which so often result in the closing down of the shop, arise in most cases from bad judgment on the part of the promoters in going into the business with only a half knowledge of the conditions to be faced. Without technical knowledge of steel making, and often without experience in the foundry business, they make mistakes in the selection of a steel-making method, of a plant location, of a type of furnace, or some such matter, that are too often fatal to the success of the shop.

CHAPTER II

GENERAL CONSIDERATIONS GOVERNING THE CHOICE OF A METHOD OF STEEL MAKING

The foundry is divided into the following departments, not all of which are included on the premises of every shop, although with the occasional exception of the last, the functions they perform are a necessary part of the production of the castings.

- 1. Steel making and raw material handling.
- 2. Pattern making.
- 3. Sand mixing.
- 4. Moulding and core making.
- 5. Casting.
- 6. Cleaning.
- 7. Annealing and heat treatment.
- 8. Finishing, straightening, welding, etc.
- 9. Laboratories.

In this chapter are set forth the general considerations governing the choice of the steel-making method to be adopted, the processes of steel making available, and the characteristic features of those processes which must be considered in making a choice among them. The processes that can be used are:

- T. Crucible.
- 2. Open-hearth, acid or basic.
- 3. Bessemer, bottom-blown or side-blown.
- 4. Electric, acid or basic.
- 5. Open-hearth, or Bessemer, and electric.

Before taking up these methods in detail, it is perhaps as well to set forth briefly here the elementary principles of steel making for the benefit of those who are not thoroughly familiar with the subject. For a really detailed exposition, the reader is referred to the standard books on Steel Metallurgy.

The raw material for all steel and iron products is iron ore, which is generally an oxide of iron, and contains about 50 per cent. metallic iron. By smelting this ore in the blast furnace with coke to provide the necessary heat and to separate the iron from the oxygen with which it is chemically combined, a product is obtained known as

pig iron. The rock with which the iron oxide is mixed in the ore is generally siliceous and infusible; limestone is charged in the furnace to unite with this rock and form a fluid slag which can be tapped from the furnace.

Four other elements besides the iron are reduced from the ore, and are absorbed by the iron in the furnace; these are silicon, manganese, sulphur and phosphorus. In addition, the pig iron absorbs from the coke a considerable content of carbon, so that the product of the blast furnace is an impure metal containing generally:

| С | Si | Mn | S | P | |
|-----------|-----------|-----------|-----------|-----------|--|
| 3-3.50 | .50-3.00 | .30-2.00 | up to .08 | up to 1 | |
| per cent. | |

Pig iron re-melted and cast into moulds is the material used in the manufacture of the ordinary grey-iron and chilled-iron castings. Depending upon the chemical composition of the metal and the rate at which it cools, the castings will be of grey iron, a comparatively soft, weak and brittle metal, or of white iron, which is intensely hard and correspondingly brittle. The fractures of the two sorts of metal, as their names imply, are either dull greyish-black, or glistening-white.

The difference in the two kinds of iron castings is due entirely to the condition of the carbon in the metal when cold. In white iron, all the carbon is chemically combined with the iron as the carbide (Fe_3C) known as cementite, which is extremely hard and brittle and gives the metal an intense hardness. In almost all cases, only certain parts of castings, such as the treads of car wheels, are purposely made white, the balance of the castings being grey (due to slower cooling), in order to give the object sufficient strength to withstand the service to which it is to be subjected.

In grey iron, much of the carbon exists in the metal in the form of graphite, which generally is in the shape of flat plates, and is distributed through the metal much as small sheets of mica would be if stirred into some plastic mass. The ground mass through which these plates of graphite are distributed is generally of about the composition of medium carbon steel; but the abundant plates of graphite so weaken the metal that is has comparatively low tensile strength and almost no ductility.

In order to convert pig iron into steel, or the parallel product, wrought iron, it is necessary to eliminate from the metal practically all the carbon, manganese and silicon it contains, at the same

time (in some processes) reducing the content of the most harmful impurities, sulphur and phosphorus. This is done by re-converting these elements to their oxides by the action on a molten bath of the impure metal, either of atmospheric oxygen, or of the oxygen of iron ore. The oxide of carbon, of course, escapes as a gas, while the silica and manganese oxide produced by the oxidation of the silicon and manganese (with which some iron oxide is unfortunately always formed and lost)unite with each other and with suitable added materials to form a fluid slag, which floats on the bath and is usually poured out of the furnace with it.

When the temperature of the furnace in which this operation is carried out is not high enough to keep the resultant bath of comparatively pure iron melted (the melting point of the iron rises as the silicon, manganese and especially the carbon are removed), the pure iron freezes a drop at a time as it is formed. The resultant product is a spongy mass of iron, its interstices filled with slag, which when rolled or hammered into bars to force out the slag and weld the particles of iron together, is known as wrought iron, or "muck bar." These furnaces are known as puddling furnaces, and the process is "pig boiling" or puddling.

At the low temperature at which puddling is carried out, considerable phoshorus is converted to its oxide and carried off in the slag, so that if the pig iron is not too high in phosphorus, a product very low in that element is obtained, which is used in crucible steel making. Puddled iron contains usually:

| С | Si | $\mathbf{M}\mathbf{n}$ | S | P |
|-----------|-----------|------------------------|------------|-----------|
| below .15 | below .15 | below .15 | .or to .o8 | .01 to .3 |
| per cent. | per cent. | per cent. | per cent. | per cent. |

Wrought iron contains also a certain amount of slag mechanically combined with the metal. It is exceedingly tough and ductile, and in its physical properties closely resembles soft steel.

When the purifying process is carried out at such a temperature that the resultant pure iron can be kept liquid in the furnace, the product is *steel*. As a rule, the bath is purified to a composition much like that of wrought iron, and a certain amount of carbon, silicon and manganese are added before the metal is poured from the furnace. Steel contains usually:

| С | Si | Mn | S | P |
|-----------|------------|-------------|-----------|-----------|
| .05 to 2 | .05 to .60 | .10 to 1.25 | .or to .r | .or to .r |
| per cent. | per cent. | per cent. | per cent. | per cent. |

The metal will be the softer and more ductile the lower the carbon,

becoming increasingly stronger and less tough as the carbon content increases. The silicon is useful chiefly to promote soundness, and the manganese partly for the same purpose, partly to off-set the action of sulphur, which tends to make the steel weak and brittle when red hot. As phosphorus causes brittleness in the steel when cold, the content of both phosphorus and sulphur must be kept very low and are often the subject of specification.

In cast steel, the carbon is rarely over .50 per cent., the silicon over .60 per cent. or the manganese over 1.50 per cent.; in fact .60 to .80 manganese is the usual composition. Cast steel of such composition, when properly made, is strong and tough, and is put to a multitude of uses for which cast iron is quite unsuited. Other things being equal, the quality of the steel made by those processes in which the bath is exposed to strongly oxidizing conditions from first to last, is lower than that of metal produced by a process in which the conditions to which it is exposed can be made less strongly oxidizing, or even reducing, when the steel is about ready to tap. The bath of fluid iron readily absorbs a certain amount of its own oxide which has a harmful influence on the properties of the metal; and although this can be partly counteracted by suitable additions to the metal in the ladle, yet the harm is not so thoroughly undone by that method as when the bath can be held under neutral or deoxidizing conditions in the furnace. When the latter procedure is possible, the oxides in the metal can be almost wholly eliminated, and the properties of the finished steel correspondingly improved.

There are, of course, a number of special or alloy steels made, containing one or more special elements, such as nickel, chrome, vanadium, etc. These elements are added usually with a view to increasing the strength and hardness without the loss of toughness that results from obtaining the desired strength by increasing the carbon. Manganese steel, in which the manganese is usually from 10 to 14 per cent., and the carbon from 1 to 1.5 per cent., is a special steel which owes its usefulness to the fact that it is at once tough, hard and very resistant to wear.

Mention should be made too, of "Malleable-iron castings," which are a product somewhat resembling steel castings and formerly sometimes sold under the guise of steel; today, however, "malleable" has found its own field of usefulness and is even extending that field at the expense of cast steel.

Malleable castings are produced by first pouring into moulds, pig iron of such composition that all the carbon it contains will be combined with iron to form "cementite" and the resultant castings show a white chill to the center; no graphite is formed in the iron as the metal cools. These castings are then annealed for several days at a fairly high temperature, to decompose the cementite into iron and carbon. The latter is deposited in the metal as fine particles of "temper carbon," which collect into rounded nodules; on account of their shape and size, these nodules do not break up the continuity of the metal, as does the flake graphite of grey iron. As the ground mass in which the carbon is distributed consists of comparatively pure iron, the castings after annealing are quite strong and ductile, generally having over 50,000 lb. per square inch tensile strength, and at least 10 per cent. extension in 2 in.

Malleable castings, therefore, are much stronger and tougher than grey iron, but not so strong and tough as well-made cast steel. They have a large field of usefulness, and will not again be referred to in this volume.

Thus the manufacture of steel and iron from the ore is carried out in two main steps; first, the production of a very impure iron in the blast furnace, then the conversion of this impure metal into the two forms of comparatively pure metallic iron known as wrought iron and steel. Both these products can be made from the ore at one step, but generally at so high a cost that the process is not a practical one.

In the crucible process, puddled iron, or open-hearth scrap low in carbon, is melted in small closed pots with sufficient charcoal or washed metal (iron containing about 3 per cent. carbon), to produce a steel of the desired carbon content. The pots contain about 100 lb. of metal each, and are heated by coal, coke, gas or oil. The process is one of pure melting, and as no impurities are eliminated from the steel, pure steel is produced only be the use of pure materials. Oxidation of the metal by the gases of the furnace is, however, largely avoided, since the steel is protected from the flame by the pot and its cover.

In the open-hearth process a shallow bath of pig iron and steel scrap is melted down in a large furnace of the bath type, in which the metal is continuously exposed to the oxidizing gases of the furnace, as the flame plays over the metal at all stages of the operation. From a bath comparatively high in silicon, manganese and carbon, these impurities are removed to the desired degree by oxidizing them, both by the action of the oxidizing gases of the furnace, and by that of a slag made oxidizing by additions of iron ore. In the acid process,

sulphur and phosphorus are not removed, so that freedom from these harmful impurities is obtained only by using very pure materials. The basic process, on the other hand, does eliminate phosphorus and sulphur, but the slags necessary for this are more oxidizing than those used in the acid process, so that the steel is constantly exposed to severely oxidizing conditions. The open-hearth furnace in foundry work is generally of about 5 to 25 tons capacity.

The Bessemer process, in American practice, removes no phosphorus or sulphur from the steel, as only acid methods are used. By this process, a molten bath of pig iron and scrap high in carbon, silicon and manganese, is converted to nearly pure iron by blowing air through the metal or upon its surface, in a cylindrical vessel containing a narrow deep bath. Carbon, silicon and manganese are oxidized and removed by the oxygen of the air blast, and the composition is then adjusted by proper additions to the molten bath. The conditions in the process are violently oxidizing, especially toward the end of the "blow," when much iron is burned by the air blast. The size of a charge, in foundry work, is from $\frac{1}{2}$ to 3 or 4 tons.

The electric furnace heats a bath of metal by means of the electric arc in a furnace much like an nropen-hearth fuace, or by means of the resistance of the metal itself to the passage of an electric current through an annular bath of small cross-section. As no oxidizing gases are introduced into the furnace, the metal is not exposed to oxidizing conditions except when desired; indeed, by proper manipulation of slag and bath, neutral or even reducing conditions are attained. In the basic process, phosphorus and especially sulphur can be reduced to extremely low percentages; and in both the basic and the acid electric furnace, not only can carbon, silicon and manganese be eliminated from the metal to the desired degree, but the metallic oxides can be reduced from the slag and metal much more thoroughly than in any other steel-making process.

In steel foundries, the electric furnace is generally used to both melt and refine the charge, but owing to the high cost of electric melting, and of removing carbon, silicon and manganese while the metal is kept hot by the electric current, it may in some instances be advisable to use the furnace only as a means of purifying openhearth or Bessemer metal. Furnaces are built in capacities of from $\frac{1}{2}$ to as high as 40 tons.

The detailed discussion of these processes is reserved for the following chapters, and here we shall consider them only in the light of their availability for our purposes, as influenced by the considerations governing the choice of a steel-making method, which may be summarized as follows:

- 1. Class, i.e., alloy steels, very pure ordinary steel, or just plain steel.

- (A) Market

 2. Variability, i.e., number of kinds of steel.
 3. Size of average casting, and of largest and smallest.
 4. Intricacy of castings.
 5. Price—largely governed by the above 1-2-3-4.
 6. Yearly tonnage in sight.
- (B) Raw material and fuel most available.
- (C) Capital available.
- (D) Competition to be faced.
- (E) Labor available.
- (F) Intermittent or steady operation.
- Ar. Quality.—Taking first the quality of the steel produced, the processes may be grouped as follows, in order of decreasing excellence, by which is meant the inherent good quality of the steel itself, unaffected by conditions of casting, or what not.
- 1. Basic electric—(alone or in connection with Bessemer or open-hearth).
 - 2. Acid electric.
 - 3. Crucible.
 - 4. Acid open-hearth.
 - 5. Basic open-hearth.
 - 6. Bessemer-side-blown.
 - 7. Bessemer—bottom-blown.

This classification is perhaps open to challenge, and the author is far from believing that it is a hard and fast one, or that it is necessarily correct under any and all circumstances. So much depends upon the raw materials used, the care and skill exercised in handling the work, the degree of refinement to which the different processes are carried, the personal equation of the operating force and the ideal of perfection adhered to by the management and held up to the men for attainment, that the excellence of the steel produced by any one method in a particular shop may be so far above or below the average for this process as to upset the classification in their case completely. In this discussion, however, are given the reasons for the above classification of the processes, assuming that in every case

the maximum possible degree of excellence is kept clearly in mind, worked for, and as nearly as possible attained.

The electric furnace clearly leads, since in no other process is it possible to hold the steel under neutral or even reducing conditions, thereby almost completely freeing it from dissolved or suspended oxides. At the same time the gases which molten steel generally holds in solution are largely eliminated, so that the metal produced is as nearly pure as we can possibly make it.

In the basic process, phosphorus and sulphur can if desired be reduced almost to traces, which is one reason for giving first place to basic rather than to acid electric steel; in the acid furnace, of course. no sulphur or phosphorus can be removed fron the metal, so that pure steel can be obtained only by using very pure raw materials. There is another reason for assigning superior quality to basic electric steel, however, which is that in the basic process the conditions to which the bath is exposed near the end of the heat can be made really neutral or even reducing. The slag at this stage of the process is a nearly pure silicate of line, and by sprinkling this slag with coke dust, or by additions of ferrosilicon, conditions in metal and slag are made reducing, and almost the last traces of oxides in the steel are converted into the metallic form. In the acid process, on the other hand, the slag is chiefly a double silicate of iron and aluminum, and it is not possible to establish reducing conditions to a sufficient extent to reduce the iron oxide content of the slag below a certain definite figure. The metal, therefore, is probably never so completely deoxidized as is well made basic electric furnace steel, and hence is of somewhat inferior quality.

Many of the makers of electric steel will not agree with this statement for which, however, good authority can be given; but probably no one to-day will quarrel with a classification that puts electric steel at the head of the list from the point of view of quality. The ease with which well-made electric furnace steel can be poured into small and intricate castings, flowing smoothly like milk, setting perfectly quietly in the sink heads, and producing castings with hardly a trace of blowholes, is largely due to its great purity.

It is well to emphasize strongly that well-made electric steel is one thing, badly and hastily made electric steel quite another. In the case of the electric furnace we have had to learn all over again that there is no cure-all for steel troubles, no panacea for all the ills to which steel is too often liable. Very great harm has been done the electric steel industry by the over-enthusiasm of some of its would-be pro-

moters, who have tried to convince the world that inexperienced men can learn overnight to make high-grade steel, simply by installing an electric furnace. So much of the steel produced by such men is of extremely poor quality that electric steel has really suffered more from its friends than from its enemies. The author once heard Capt. Robert W. Hunt state in discussion "The only way to make good steel is to make good steel," and that somewhat cryptic statement has proved to be as true of the electric furnace as of any of the older processes.

The crucible process produces excellent steel, the finest that could be made, up to the date of the introduction of the electric furnace. Some authorities may even deny that the electric product excels that of the crucible. The excellencies of crucible steel are due to the protection of the melting and molten steel by the pot and its cover from the oxidizing gases of the furnace, so that after the oxygen of the air in the pot is exhausted by the burning of charcoal used as a source of carbon, or by the oxidation of the carbon, silicon and manganese of the metal, the steel under its layer of slag is exposed to a practically neutral atmosphere. Moreover, silicon added purposely. or absorbed from the clay of the pot, deoxidizes the steel. Thus an excellent product is obtained, even if the raw material is merely steel scrap from the open-hearth process. If the raw material be puddled iron, for reasons which will be set forth at greater length later, a much better steel is the result. But whatever be the raw material. the excellence of the steel is due to the refining in the neutral or slightly reducing atmosphere of the pot, of metal that has previously been subjected to strongly oxidizing conditions, either in the puddling furnace, the open-hearth furnace or the Bessemer vessel.

Though there is evidence tending to show that the metal is freed from oxides to a considerable extent in the puddling furnace, and that the puddled metal is further improved in crucible melting; yet it is impossible to conceive that with the surely moderately reducing conditions in the pot, even puddled metal, much less open-hearth steel, can be freed from oxides to as great an extent as is possible in the electric furnace. We believe that with the greatest care used in both processes, electric steel will excel crucible steel, and that on the average electric steel far outstrips the crucible steel generally made for castings, using open-hearth steel scrap as raw material.

That crucible steel is superior to open-hearth steel is widely accepted, and some of the reasons for that superiority are suggested

by the foregoing discussion. In the open-hearth furnace the metal is subjected to strongly oxidizing conditions throughout the process, and the means at the disposal of the steel maker for reducing the oxides of iron absorbed by the steel are insufficient to carry this deoxidation to anything like completion. Hence the quality of the steel produced is inferior to that of crucible or electric furnace steel.

In acid open-hearth practice, no phosphorus and sulphur are removed from the steel, and there is even a slight gain in these impurities, while in the basic furnace the conditions are such that phosphorus and sulphur are largely eliminated; yet more strongly oxidizing slags are used in the basic furnace, and in order to facilitate the removal of phosphorus and sulphur the steel is often refined down to a very low content of carbon, which makes it more easily oxidized, and hence promotes the absorption of oxides. Add to this the fact that very impure grades of scrap are largely used in the basic furnace, and that the deoxidizing additions are generally so used that their effect is much less complete than in acid practice, and we have an array of considerations that point clearly to the superiority of acid over basic open-hearth steel. In some shops it has been found that this superiority can be measured roughly in terms of strength, and that for equal strength basic steel must have 5 "points" more carbon than acid steel.

We now come to the discussion of the excellence of Bessemer steel, and its classification relative to open-hearth. Here the author confesses he is in doubt, and hesitates to classify Bessemer steel as made for castings, below basic open-hearth. After long deliberation, the following discussion appears to him reasonable.

In Bessemer steel making air is blown through a bath of iron containing much carbon, silicon and manganese, or a blast of air is projected strongly upon the surface of such a bath. By the oxidizing action of the air, the carbon, silicon and manganese are converted to CO (gas), and to SiO₂ and MnO, the first escaping, the last two going to form slag. That much iron must at the same time be oxidized where the excess of air is great, especially at the end of the blow when the iron is no longer protected from such oxidation by carbon, silicon and manganese, is axiomatic, and is conclusively proved by the considerable loss of iron in conversion. That the oxidation by the blast is greater than the oxidation in the openhearth furnace by the gases and the oxidizing slag, is undeniable. At the end of the blow, after the maximum oxidation of iron has taken place, the vessel is at once turned down, containing a steel very

rich in oxides and gases (as shown by its "wildness," if we try to pour it at once), and we add ferrosilicon and ferromanganese to reduce the oxides. In the open-hearth process the oxidizing conditions grow less severe as the process proceeds, and we allow the steel to boil for some time, during which it undoubtedly frees itself from part of the oxides absorbed in the earlier stages of the process.

Open-hearth steel is, therefore, purer than converter steel, before the deoxidizing additions are made; and if these additions are used in such a way that their effect is as complete in the one case as in the other, the superiority of the open-hearth steel will be maintained. Since open-hearth steel in foundry work is generally poured into larger ladles than is Bessemer steel, solid additions in the ladle have longer to act, and hence do their work more thoroughly, in openhearth practice. When the additions are made to the open-hearth furnace, they deoxidize the steel more completely than similar additions made to the Bessemer converter, because they have longer to act. But liquid additions to the Bessemer vessel or ladle probably do their work more thoroughly than solid additions to ladle or furnace. It is, of course, possible to use liquid additions in open-hearth practice, but it is seldom done, and in fact the additions are quite generally made in the ladle. Bessemer foundrymen, on the other hand, usually prefer to make their additions to the vessel, and many of them use melted material.

Open-hearth metal, therefore, can be made superior to Bessemer, but there is no doubt that owing to the more complete deoxidation often attained in Bessemer practice, much of the open-hearth steel poured in foundries is actually inferior to the best Bessemer steel made for similar purposes.

It is difficult, perhaps impossible to determine whether the small bottom-blown or the side-blown Bessemer converter produces the better steel; indeed in this second edition the author has reversed his judgment of 7 years ago, and given the preference to side-blown metal. It would seem at first glance as if air blown through the metal must oxidize more iron than air blown upon the surface of the metal. However, partly because the higher temperature of side-blown steel promotes oxidation of the iron and absorption of the oxide by the steel, partly because in the bottom-blown vessel, the iron oxide formed at the mouths of the tuyères is largely reduced again by the carbon of the overlying bath, actually the loss of iron is considerably greater in the side-blown vessel than in the bottom-blown. It would appear, therefore, that the metal must be less

pure where the greatest oxidation has occurred and the greatest opportunity given for absorption of the oxide, and that other things being equal the superior position should be assigned to bottomblown steel.

Yet it appears to be a fact that it is easier to "quiet" side-blown metal and that it takes less silicon, manganese and aluminum to make side-blown steel set sound in the moulds than must be used for bottom-blown steel. Moreover, tests of side-blown converter metal produced in the best shops are at least the equal of tests of similar metal made in the bottom-blown vessel. Possibly the steel in the bottom-blown vessel absorbs more nitrogen from the air blast than side-blown metal does, or some other factor we do not understand may account for the fact that side-blown metal appears to be less "wild" than bottom-blown. The side-blown vessel, it should be said, is almost exclusively used in the steel foundry, and in numerous shops steel of very high grade is produced.

Before leaving the subject of the quality of the steel produced by the different processes, it is well to add that the choice of a process is largely governed by other considerations than the inherent quality of the steel produced. Well-designed steel castings, made by any of the processes carefully conducted, when free from flaws and blowholes and properly annealed or heat treated are excellently adapted to the uses to which they are put and will show by tests that they possess strength and ductility far beyond what they ever will be called upon to exhibit.

There is another angle, however, to this question of the quality of the steel of which the castings are made, which is the effect of the average quality of the steel upon the soundness of the castings, in other words, the uniformity of excellence, as it were, that may be expected of the processes. The foregoing discussion is based upon the assumption that the steel in every case is made with sufficient care and skill so that it will not liberate gases in solidifying in the moulds, with the result that the castings show more or less numerous blowholes. By all the processes it is possible to produce such steel, but of course at times something may go wrong, with the result that the steel proves to be "wild" in the moulds, and spongy castings are the result.

In the opinion of the author, the processes may be grouped in the same order from this point of view as from that of the inherent quality of the steel itself. Basic electric steel, skilfully made, is probably the least liable to these occassional poor heats, and bottom-blown

converter steel the worst, the other processes coming in the order given for quality.

The worst of this matter of occassional "wild" heats is that frequently the foundryman has no warning of the coming trouble until several castings have been poured and have partly solidified. The steel often appears to be all right and pours all right, and when the tops of the risers freeze over apparently nothing is amiss; then without warning gas is liberated in the solidifying castings in sufficient volume to cause the liquid metal within the risers to break through the crust and begin to bubble up and run over the moulds, producing a "cauliflower head." Aluminum must then be used to quiet the steel for the rest of the castings poured from the heat, but some are liable to be unsound, and all suffer from overdoses of aluminum.

The source of the gases liberated in the solidifying steel which gives rise to this trouble is in most cases probably oxide of iron in solution in the metal. The reaction of this oxide with the carbon of the recarburizers produces carbon monoxide gas (CO); a great deal of it is probably generated while the steel is still in the ladle, and escapes. Some of it, however, together probably with nitrogen and other gases (especially in the converter process) appears to be dissolved, and to be liberated later in the cooling metal with disastrous results.

There are, of course, many other causes of blowholes than "bad steel" which are discussed later, and the buyer of castings must not conclude that the steel is necessarily of poor quality because he gets some porous castings. Most foundrymen are familiar now with the experience of one plant using the basic electric process for the production of ordnance castings during the war; though probably the most skilful makers of electric steel in America went to this plant and made steel, yet a tremendous percentage of the castings were full of blowholes, and the trouble was finally found to be entirely with the sand used for the moulds.

A2. Variability of Product.—This demands flexibility in the steel-making process. By this is meant the ability to produce readily steels of widely differing analyses to take care of orders for small lots of steel of special composition. Such orders form a very inconsiderable proportion of the steel foundry business, and in almost no case can be counted upon as a steady source of income-producing work. As in some special cases this may be a consideration worth taking into account, it must be touched on briefly here.

In order of decreasing flexibility, the processes are:

- 1. Crucible.
- 2. Electric.
- 3. Bessemer.
- 4. Open-hearth.

The flexibility is, of course, determined by the weight of metal in each lot melted and by the difficulty of dividing a lot into parts. On the score of weight of lots, the flexibility is therefore in the order of capacity; but the capacity in the case of crucible melting is not that of the whole furnace, but that of a pot. In crucible melting, of course, each pot is a law unto itself, so that within limits as many different steels may be melted at once as there are pots in the furnace. Indeed, as we shall see, the difficulty in crucible work is just the other way around, and great efforts must be made to produce uniform steel. There are certain sorts of steel, such as very low carbon, that are hard to produce from crucibles, but the process is out and away the most flexible.

The contents of an electric furnace may be brought to a certain composition, enough steel of this sort poured off for the purposes in hand, and the composition of the bath then changed to suit other requirements. It is easy to see that only the small furnace lends itself to such treatment, as the expense of so handling 10 or 15 tons would sometimes be prohibitive. Or, we may produce steels of different compositions from a uniform bath by making our additions in the ladle. There are two objections to this method, which are first, that it sacrifices the benefit of thoroughly mixing our steel in the furnace, and second, that it is difficult to weigh accurately the amount of steel poured out, and so to secure correct composition. Neither of these methods is wholly satisfactory, and except when absolutely necessary, it will be found to be the best practice not to attempt to "split heats."

In Bessemer work, after the steel is blown no further heat is added to it, and it begins to cool as soon as the vessel is turned down. Therefore, the only method of making different compositions of steel from one blow is to do the mixing in the ladle. The same considerations apply to this as to doing the same thing with an electric furnace, and in practice the method is rather unsatisfactory. The chief difficulty is the slag, which, if fluid, cannot be held back in pouring, and makes it practically impossible to pour the proper quantity of metal into each ladle. As the metal cools quite rapidly, it must be handled without much delay. For this reason, not over

two kinds of steel can be made from one heat. The Bessemer process is, therefore, less flexible than the electric.

In open-hearth work, it is far from easy to make more than one kind of steel at a time. The furnaces being of large capacity it is not commercially practicable to make different compositions successively in a tilting furnace; and the steel rushes so rapidly from fixed furnaces that even with a bifurcated spout the accurate proportioning of steel to each of two ladles containing different recarburizers is extremely difficult. Should the furnaces be tilting, not fixed, more can be done in this line, but even then we have the slag to hamper us and in the one shop which to the author's knowledge follows this practice, the men in charge admit that it is awkward and would not recommend a new concern to copy their installation. To transfer steel of low carbon from one ladle to another of smaller capacity in which further additions are made, and so produce two steels at one tap is sometimes practicable, but the limit of such procedure is soon reached by the cooling of the steel. At least one foundry, however, makes a small lot of manganese steel from an occasional heat of carbon steel by this method.

A₃-4. Size and Intricacy.—The processes in order of decreasing suitability for pouring small and intricate work can be tabulated as follows:

- 1. Crucible.
- 2. Electric.
- 3. Side-blown Bessemer.
- 4. Bottom-blown Bessemer.
- 5. Acid open-hearth.
- 6. Basic open-hearth.

This classification is a difficult one to fix arbitrarily, since there are many factors in the problem to be taken into consideration, which in particular cases may shift the relative positions of the processes.

To take a single instance, electric steel, owing to its great purity and freedom from gas, and to the high temperature at which it may be poured, is eminently suited to pouring castings of the very lightest weight and of very thin section. Many excellent castings that have been regularly poured of electric steel would be almost impossible to produce by the older processes. Should the electric furnace that is being used be of small size, therefore, the electric furnace may well handle a certain proportion of work that even the crucible foundryman would hesitate to tackle.

On the other hand, the crucible process is pre-eminently one of

small units. The average pot contains from 90 to 100 lb. of steel. and it has never proved economical to melt in larger crucibles. steel can be made exceedingly hot, fluid, and free from gas, runs well in very light sections, and is kept hot by the hot pot, a very poor conductor of heat and so shaped as to present a very small surface of liquid steel compared to its mass. When the practice of pouring from the pots is followed, the steel can be brought to the moulds very hot, and all poured before it has an opportunity to chill. It would have to be a very small electric furnace indeed that would make it economical or possible to put all the steel into very light castings at the maximum temperature. If the steel be poured directly from a large ladle it is impossible to do good work with very small castings, besides which the steel in the ladle would soon chill, at the very slow rate at which it would be disposed of. To "shank" from the large ladle manifestly would not bring the steel, especially that from the last of the ladle, to the moulds at as high a temperature as that of the pots kept hot in the crucible furnace until wanted. shank from the furnace is possible, but often most unsatisfactory. Lastly, the cost factor enters into this question very strongly, since the smaller electric furnaces cost more to operate per ton of steel than the large ones, the cost increasing in inverse ratio to the size. Hence to pour exclusively very light work, using a number of small electric furnaces, considerably increases the cost of the steel in the ladle over what is possible with furnaces of 3 to 6 tons. In the crucible process, on the other hand, the unit of weight per pot is fixed and does not enter into the problem of cost of steel, which is determined by the size and style of furnace used. The cost per pound of steel decreases with the increase in number of pots melted per day, and the process is as well suited to a large production of very light work as to a small tonnage. The leading position is, therefore, given to the crucible process, with the qualifying statement that other considerations may in particular cases put the electric furnace in the lead.

Again, the advocates of the small Bessemer converter, especially of the side-blown vessel of about 2 tons capacity, point out that their process is in many ways excellently adapted to the production of the very lightest work. The temperature of the metal can be made very high, the total tonnage of steel to be disposed of per blow is low, and by keeping the steel hot in the vessel and pouring from shanks, or pouring from a ladle with a number of shanks, extremely light castings can be poured. To this we may answer that

as compared with the crucible furnace, the steel will undoubtedly chill faster in the unfired, although hot, vessel or ladle, than in the furnace, so that the last of the heat will often be colder in Bessemer practice than in crucible work, in spite of its initial extreme heat. Coupled with this is the fact that Bessemer steel is necessarily very full of oxides and gases, and hence "wild," when the vessel is turned down, and the time allowed for the action of the deoxidizers being short, the steel is never as pure, gas free and smooth flowing as crucible or electric steel. We are compelled, therefore, to conclude that in its nature the Bessemer process is less suitable to the production of a large tonnage of very light work than either the crucible or the electric process. It is at its best for the production of castings weighing from 25 to 200 or 300 lb.

The side-blown Bessemer vessel as a general rule produces much hotter steel than the bottom-blown, and hence can be used for the manufacture of lighter castings. In some shops, where the bottom-blown vessel is run by men of exceptional skill and experience, it has been successfully used for light work; as commonly handled, however, the side-blown vessel does better work on light castings than the bottom-blown.

The open-hearth furnace is not at all suited to light work. This is due to the generally large tonnage produced at a heat, and to the fact that the whole heat is from the necessities of the case commonly poured at once. Both of these factors militate against the pouring of large numbers of small castings, since the steel would be seriously chilled before all of it could be run into light work. The use of nozzles to pour with, and the necessarily large and violently flowing stream involved, prevents the proper pouring of small castings. Moreover, the steel is not so hot as that produced by the other processes and is less pure and well killed than crucible or electric steel. For castings averaging over 200 lb., on the other hand, the open-hearth furnace is the most suitable of the processes, especially as this heavy work commonly requires only a good average grade of steel and there is no incentive to the use of higher grade materials.

That basic open-hearth steel is less well killed, more oxidized, and contains more gas, and is therefore "wilder" and more difficult to pour into light work than acid open-hearth steel, is in the opinion of the author widely accepted, and certainly is in line with theoretical considerations of the case.

Some years ago a special design of small open-hearth furnace (W. M. Carr's combined furnace and ladle), came into notice, and ex-

ceedingly good work was done with these furnaces in pouring very pure metal into light castings. Other small open-hearth furnaces, such as the McLain-Carter design have also been much advertised, and installed in a number of shops. We doubt, however, if the experience of the users of these small furnaces for the production of light castings justifies a rearrangement of this classification in the case of the open-hearth furnace. A foundryman of long experience informed the author that in one shop using both side-blown converters and a small open-hearth furnace, he noticed that metal from the converters was often taken into the open-hearth shop, and vice versa, and although the men in charge had assured him that the open-hearth furnace was excellent for light work, yet in practice they went to the trouble of transferring the ladles from one foundry to another in order to pour the heavier work with the open-hearth steel and the lighter work with converter metal.

A5. Price of Castings.—The price paid for castings of a given sort of steel is largely governed by the size and intricacy of the castings, since this involves the amount of moulding and core making time spent upon the mould per pound of casting, the difficulty of securing good castings, etc. Though special prices are paid for alloy steels, that of ordinary carbon-steel castings, except in special cases does not at present depend upon quality of steel as such. The desideratum is a casting as free as possible from blow holes, cracks and flaws and other defects, of a steel well enough made and annealed to pass the usual specifications. Since the tests commonly specified are intended to exclude only badly made and badly annealed steels. they are comparatively easy to pass with steel made by any of the commercial processes. With a comparatively few exceptions, therefore, superior quality, whether due to the steel-making methods used, or to the heat treatment put upon the steel, will not ensure higher prices and the more costly processes should be adopted only when they are absolutely needed to produce sound castings from the patterns to be handled. It should be remembered, however, that the cost of the steel is not the only factor in the cost of the castings.

The processes are grouped as follows, in order of increasing cost of steel, the estimates being based upon prices of raw materials in the Philadelphia market as given in the *Iron Age* for June 30, 1921.

- I. Basic open-hearth.
- 2. Acid open-hearth.
- 3. Acid electric.
- 4. Side-blown Bessemer.

- 5. Basic open-hearth and electric furnace.
- 6. Bottom-blown Bessemer.
- 7. Basic electric.
- 8. Gas crucible furnace, 30 pot.
- 9. Crucible furnace, coal holes or oil melting.

The variations in the price of raw material, fuel, power, etc., from time to time and from place to place, and the effect of fluctuating output, make these figures of value chiefly to illustrate the factors governing the costs of production. Some of the estimates, for instance that for the electric furnace, are considerably higher than many that have been published. In figuring these costs, an effort has been made to adhere as closely as possible to the conditions of practice in a foundry, where it is not possible to secure the economies easily attained in large shops equipped with a number of furnaces, and not subject to delays, wide variations in output, etc. The figures here given are intended to represent conservative estimates of cost, that will not often be exceeded, rather than the much lower figures that can be attained when the process is being run under more nearly ideal conditions.

It is just as well to state here that a great many of the so-called estimates of cost of steel that have been published by the promoters of various types of electric furnaces are utterly misleading, and should not be taken seriously. Electric-furnace promoters have given the author figures for electrode consumption, for instance, that proved to be about one-third the figure of one of the best electric-furnace operators in the country, and the same is true of the other items in these delightful works of fiction so often published as sober fact.

Costs of steel per gross ton in a 15-ton basic open-hearth furnace, burning producer gas and making 15 heats per week, or 225 gross tons.

RAW MATERIAL, assuming a conversion loss of 5 per cent., and using 40 per cent. pig, 30 per cent. purchased scrap, and 30 per cent. shop scrap.

Fifteen tons \div .95 = 15.79 tons, of which .135 will be ferros, leaving 15.655 tons, of which

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6.262 tons will be pig,
4.6925 tons will be purchased scrap,
4.6925 tons will be shop scrap.
6.262 tons basic pig at $24.00, = $150.29
4.6925 tons purchased scrap at $11.00, = 51.62
4.6925 tons shop scrap at $15.00, = 70.39
.075 tons ferromanganese at $80.00 = 6.00
.06 tons ferrosilicon at $70.00, = 4.20

$282.50 or for 15 tons
$18.83 per ton
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LABOR PER WEEK

2 melters at \$200 per month, = \$93.02 2 second helpers at \$6.00 per day, = 72.00 2 ladle men at \$6.50 per day, = 78.00 2 gas makers at \$5.40 per day, = 64.80 4 common laborers at \$4.20 per day, = 100.80

Production per week 15 × 15 = 225 tons. $\frac{408.62}{225}$ = \$1.82 per ton

| | Per ton |
|----------------------------------|--------------|
| Summary | steel tapped |
| Raw material | \$18.83 |
| Labor | 1.82 |
| Coal, 700 lb. at \$7.00 | 2.45 |
| Yard labor | 45 |
| Repairs and maintenance | 1 . 50 |
| Supplies and miscellaneous | 1 . 50 |
| Management 50 per cent. of labor | 91 |
| | - |
| | \$27.46 |

Interest and depreciation, 15 per cent. on \$100,000 = \$15,000 per year. At 225 tons per week, 48 weeks per year, we produce 10,800 tons of steel. This item then amounts to about \$1.40 per ton, bringing our total cost to \$28.86 per gross ton.

Estimating 60 per cent. good castings, 40 per cent. scrap, we have \$28.86, less credit for .40 tons scrap at \$15.00 per ton (\$6.00) = \$22.86, cost of metal for .60 tons good castings; $\frac{22.86}{.60}$ = \$38.10 per gross ton of good castings.

W. M. Carr gives in his book, "The Manufacture of Open-hearth Steel Castings," the following figures on the cost of installation of open-hearth plants;

| Ітем | | Cost of installation per ton of steel | |
|------|-------------------------------|---------------------------------------|--|
| | Basic furnace. | \$1200 | |
| | Gas producers, etc | 600 | |
| | Power, machinery, cranes, etc | 2300 | |
| | | | |
| | | \$4100 | |

This figure has been raised about $66\frac{2}{3}$ per cent. to allow for the higher costs prevailing to-day.

On this basis, a 15-ton furnace will cost \$100,000, not including buildings.

Costs of steel per gross ton in a 15-ton acid open-hearth furnace, burning producer gas and making 16 heats per week or 240 tons.

RAW MATERIAL, assuming a conversion loss of 4 per cent., and using 20 per cent. pig, 30 per cent. shop scrap and 50 per cent. purchased scrap.

Fifteen tons \div .96 = 15.63, of which .22 tons will be ferros, leaving 15.41 tons, of which

\$337.16 or \$22.48 per ton

LABOR

$$408.62 \div 240 = 1.70 per ton

| Summary | Per ton steel tapped |
|----------------------------------|-------------------------|
| Raw material | |
| Labor | |
| Coal | |
| Yard labor | |
| Repairs and maintenance | 1.20 |
| Supplies and miscellaneous | 1.20 |
| Management 50 per cent. of labor | 85 |
| | \$30.33 |

Interest and depreciation, 15 per cent. on \$95,000 = \$14,250 per year. At 240 tons per week, 48 weeks per year, we produce 11,520 tons of steel. This item then amounts to \$1.24 per ton, bringing our total cost to \$31.57 per ton of steel in ladle.

Estimating 60 per cent. good castings, 40 per cent. scrap, we have \$31.57, less credit for .40 tons scrap at \$14.00 per ton (\$5.60)

- = \$25.97, cost of metal for .60 tons good castings; $\frac{$25.97}{.60}$ = \$43.28 per gross ton of good castings.
- W. M. Carr (loc. cit.) gives cost of installation of acid open-hearth plants, not including building, as follows:

| | n method of steel making | Y G | 21 |
|----------|--|------------------|-----------------------------|
| Item | | installation | on per |
| A | Acid furnace | \$1000 | |
| (| Gas producers, etc | 600 | |
| I | Power, machinery, cranes, etc | 2300 | |
| | | | for a 15-ton e, \$58,500 |
| This | s figure has been raised for estimating purp | oses abou | ıt 66∦ per |
| cent. | -8 | | 8 P |
| | ts of steel per gross ton in a 3-ton botton | m-blown | Bessemer |
| vesser, | making 20 heats per day or 60 tons. | | |
| using 6 | MATERIAL, assuming a melting and conversion loss oper cent. pig, 30 per cent. shop scrap, 10 per cent. et tons \div .88 = 3.41 tons, of which .05 tons will be | purchased | l scrap. |
| tons, of | | | |
| | 2.016 tons will be pig, | | |
| | 1.008 tons will be shop scrap, | | |
| | o. 336 tons will be purchased scrap. | e -c c- | |
| | 2.016 tons low phosphorus pig at \$38.00 1.008 tons shop scrap at \$14.00 | \$ 76.6 1 | |
| | o. 336 tons low phosphorus scrap at \$18.00 | 6.05 | |
| | 0.024 tons ferrosilicon at \$70.00 | 1.68 | |
| | 0.026 tons ferromanganese at \$80.00. | 2.08 | |
| | | | or \$33.51 per |
| Labor | | Per day | |
| LABUR | Charging cupola and yard, 6 men at \$3.50 | 1 cr day | |
| | • | \$21.00 | |
| | r cupola tender at \$5.00 per day | 5.00 | |
| | r cupola tender helper at \$4.00 per day. | 4.00 | |
| | 6 vessel men at \$4.50 per day | 27.00 | |
| | 1 blower at \$8.00 per day | 8.00 | |
| | \$ | \$65.00 or \$ | 1.08 per ton |
| | | Per to | n of steel in |
| Summai | RY | | ladle |
| R | aw material | \$ | 33.5 r |
| | abor | | 1.08 |

| UMMARY | ladle |
|--------------------------------------|-----------------|
| Raw material | \$33.5 r |
| Labor | 1.08 |
| Fuel, \frac{1}{5} ton coke at \$7.25 | 1.45 |
| Repairs and maintenance | .80 |
| Supplies and miscellaneous | . 50 |
| Power | 3 · 75 |
| Management 50 per cent. of labor | . 54 |
| | \$41.63 |

Interest and depreciation, 15 per cent. on \$60,000 = \$9000 per year. At 300 days per year, 60 tons per day, we produce 18,000 tons of steel. This item then amounts to \$.50 per ton, bringing our total cost to \$42.13 per gross ton of steel in ladle.

Estimating 60 per cent. good castings, we have \$42.13, less credit for .40 tons scrap at \$14.00 per ton (\$5.60) = \$36.53, cost of metal for .60 tons good castings; $\frac{$36.53}{.60}$ = \$60.88 per gross ton of good castings.

Costs of steel per gross ton in two 2-ton side-blown Bessemer vessels, making 12 heats per day, or 24 tons.

RAW MATERIAL, assuming a melting and conversion loss of 16 per cent., and using 50 per cent. pig, 30 per cent. shop scrap and 20 per cent. purchased scrap.

```
Two tons \div .84 = 2.38 tons, of which .034 tons will be ferros, leaving 2.346
tons, of which
                1.173 tons will be pig,
                 . 7038 tons will be shop scrap,
                 .4692 tons will be purchased scrap.
      1.173 tons low phosphorus pig at $38.00.... $44.57
       . 7038 tons shop scrap at $14.00.....
                                           9.85
       .4692 tons low phosphorus scrap at $18.00.
                                           8.45
       .016 tons ferrosilicon at $70.00: . . . . . . . . .
                                           I.12
       .018 tons ferro manganese at $80.00.....
                                           I.44
                                         $65.43 or $32.72 per
                                                 ton
LABOR
                                              Per day
         Charging cupola and yard, three men at $3.50
           per day..... $10.50
         1 cupola tender at $5.00 per day.....
         3 vessel men at $4.50 per day.....
                                              13.50
         1 blower at $8.00 per day.....
                                              8.00
                                             $37.00 or $1.54 per
                                                   ton
                                           Per ton of steel in
SUMMARY
                                               ladle
    Raw material.....$32.72
    Labor....
                                                 1.54
    Fuel ton coke at $7.25.....
                                                1.45
    Repairs and maintenance.....
                                                1.00
    Supplies and miscellaneous.....
                                                 . 50
    Power....
                                                 .50
    Management 50 per cent. of labor.....
                                                 .77
                                               $38.48
```

Interest and depreciation, 15 per cent. on \$43,000 = \$6450 per year. At 300 days per year, 24 tons per day, we shall produce 7200 tons of steel. This item then amounts to \$.90 per ton of steel, bringing our total cost to \$39.38 per gross ton of steel in the ladle.

Estimating 60 per cent. good castings, we have \$39.38, less credit for .40 tons scrap at \$14.00 per ton (\$5.60) = \$33.78, cost of metal for .60 tons good castings; $\frac{$33.78}{.60}$ = \$56.30 per gross ton of good castings.

Costs of steel per gross ton in a 3-ton basic electric furnace, melting cold scrap, working double turn and making 30 heats per week.

RAW MATERIAL

| .60 ton high phosphorus scrap at \$10,00 | \$ 6.00 |
|--|------------|
| .40 ton shop scrap at \$18.00 | \$ 7.20 |
| .008 ton ferromanganese at \$80.00 | \$.64 |
| .008 ton ferrosilicon at \$70.00 | . 56 |

\$14.40 per ton

Per week

Labor

| 2 | melters a | at | \$200.00 per month | \$ 90.30 |
|---|-----------|----|--------------------|-------------|
| 2 | helpers a | ιt | \$6.00 per day | 72.00 |

4 helpers at \$4.20 per day.....

\$263.10 or for 90 tons \$2.92 per ton

Per ton of steel in ladle

SUMMARY

| Raw material | \$14.40 |
|----------------------------------|---------|
| Labor | 2.92 |
| Power, 800 kwhr. at .02 | 16.00 |
| Electrodes | 2.50 |
| Repairs and maintenance | 2.00 |
| Supplies and miscellaneous | 1.50 |
| Yard handling | . 50 |
| Management 50 per cent. of labor | r.46 |

\$41.28

Interest and depreciation, 15 per cent. on \$60,000 = \$9000 per year. At 90 tons per week, 48 weeks per year, we produce 4320 tons of steel. This item then amounts to \$2.09 per ton, bringing our total cost to \$43.37 per gross ton of steel in the ladle.

Estimating 60 per cent. good castings, we have \$43.37, less credit for .40 tons scrap at \$18.00 per ton (\$7.20) = \$36.17, cost of metal for .60 tons good castings; $\frac{$36.17}{.60} = 60.28 per gross ton of good castings.

Costs of steel per gross ton in 3-ton basic electric furnace, refining molten Bessemer or open-hearth metal, and making 70 heats per week or 210 tons.

| Labor | | Per week |
|-------|------------------------------|-----------|
| | 2 melters at \$200 per month | . \$90.30 |
| | 2 helpers at \$6.00 per day | . 72.00 |
| | 2 helpers at \$4.20 per day | . 50.40 |
| | | |

\$212.70 or for 210 tons, \$1.02 per ton

| Summary | Per ton of steel in ladle |
|----------------------------------|---------------------------|
| Labor | \$1.02 |
| Power, 400 kwhr. at .02 | 8 . 00 |
| Electrodes | I.25 |
| Repairs and maintenance | I.00 |
| Supplies and miscellaneous | 0 . 75 |
| Management 50 per cent. of labor | |
| | \$12.53 |

Interest and depreciation, \$9000 per year. At 210 tons per week, 48 weeks per year, we produce 10,080 tons of steel. This item then amounts to \$.89 per ton, bringing our cost to \$13.42 per ton. This is to be added to the cost of open-hearth or Bessemer metal in the ladle.

In the case of Bessemer metal, the credit for shop scrap, and also the value of scrap charged to the vessel, should be taken on a basis of about \$18.00 per ton of scrap, because low phosphorus steel will be made in the electric furnace.

Costs of steel per gross ton in a 3-ton acid electric furnace, melting cold scrap, working double turn and making 55 heats per week or 165 tons.

RAW MATERIAL

| .60 tons low phosphorus sc | rap at S | \$18.00 | \$10.80 |
|----------------------------|----------|---------|---------|
| .40 tons shop scrap | at | 14.00 | 5.60 |
| .008 tons ferromanganese | at | 80.00 | . 64 |
| .oı tons ferrosilicon | at | 70.00 | . 70 |
| | | | |

\$17.74

| Labor | | Per week |
|-------|---------------------------------|----------|
| | 2 melters at \$200.00 per month | \$90.30 |
| | 2 helpers at \$6.00 per day | 72.00 |
| | 6 helpers at \$4.20 per day | 151.20 |

^{\$313.50} or for 165 tons \$1.90 per, ton

| SUMMARY | r ton | of steel in ladle |
|----------------------------------|-------|-------------------|
| Raw material | | \$17.74 |
| Labor | | |
| Power 500 kw. at .02 | | 10.00 |
| Electrodes | | 1.75 |
| Repairs and maintenance | | 1.50 |
| Supplies and miscellaneous | | |
| Yard handling | | |
| Management 50 per cent. of labor | | .95 |
| | | \$36.34 |

Interest and depreciation, \$9000 per year. At 165 tons per week, 48 weeks per year, we produce 7920 tons of steel. This item then amounts to \$1.13 per ton, bringing our cost to \$37.47 per ton.

Estimating 60 per cent. good castings, we have \$37.47, less credit for .40 tons scrap at \$14.00 per ton (\$5.60) = \$31.87, cost of metal for .60 tons good castings; $\frac{31.87}{60}$ = \$53.11 per gross ton of good castings.

Costs of steel per gross ton in a 20-pot crucible steel plant, using coal holes and making 2 heats per day. Pots containing 100 lb. of steel each, bringing our output to 4000 lb. of steel per day. Pots will average 3 heats each.

RAW MATERIAL

| 59 lb. low phosphorus boiler plate punch- | |
|---|---------|
| ings at \$17.00 | \$.4031 |
| 1 lb. washed metal at \$30.00 | .0134 |
| 40 lb. shop scrap at \$17.00 | . 3035 |
| ½ lb. ferromanganese at \$80.00 | .0178 |
| ½ lb. ferrosilicon at \$70.00 | .0155 |
| | |

\$.7533 or \$.007533 per lb.

| т | | _ | _ | _ | |
|----|---|---|---|---|--|
| 14 | А | В | u | ĸ | |

| | _ |
|--------------------------------|---------|
| 1 melter at \$12.00 per day | \$12.00 |
| 1 pot puller at \$8.00 per day | 8.00 |
| ı helper at \$5.00 per day | 5.00 |
| ı helper at \$4.50 per day | 4.50 |

\$29.50 or

Per day

\$.007375 per lb.

Pots

1 pot worth \$5.00 makes 300 lb. of steel. Pots, therefore, amount to .016666 per pound of steel.

| Summary | Per pound of steel |
|--------------|---------------------------------|
| Raw material | |
| | \$.037262 or \$83.47 per ton |
| Items above | |
| | \$126.47 per ton of steel |

Interest and depreciation, 15 per cent. on \$10,000, amounts to \$1500 per year. At 4000 lb. of steel per day 300 days per year, we shall produce 536 tons per year. This item therefore amounts to about \$2.80 per ton, bringing our total cost to \$129.27 per gross ton of steel in the pots.

Estimating 60 per cent. good castings, we have \$129.27, less credit for .40 tons scrap at \$17.00 per ton (\$6.80) = \$122.47 cost of metal for .60 tons good castings; $\frac{122.47}{.60}$ = \$204.10 per gross ton of good castings.

Costs of steel per gross ton in a 30 pot regenerative crucible furnace, burning producer gas, and making 6 heats per 24 hours. Production 18,000 lb. per day. Pots will average $3\frac{1}{2}$ heats each.

| Labor | | Per week |
|-------|---------------------------------|----------|
| | 2 melters at \$12.00 per day | \$144.00 |
| | 4 pot pullers at \$8.00 per day | 192.00 |
| | 2 helpers at \$5.00 per day | 60.00 |
| | 2 gas men at \$5.40 per day | 62.40 |
| | 2 gas men at \$4.20 per day | 50.40 |
| | | |

\$508.80 or for 108,000 lb. of steel, \$.004711 per lb.

in pot.

Pots

¹ pot worth \$5.00 makes 350 lb. of steel. Pots, therefore, amount to \$.014286 per pound.

| SUMMAR | Per pound of steel |
|--------|---|
| | Raw material (as before) \$.007533 Labor .004711 Pots .014286 Management 50 per cent. of labor .002355 |
| | \$.028885 per pound or \$64.70 per ton Per ton of steel |
| | Items above \$64.70 Repairs and maintenance 2.00 Fuel, 1500 lb. gas coal at \$7.00 per ton 5.25 Supplies and miscellaneous 1.50 Power .40 |
| | 2 72 2 7 |

Interest and depreciation, 15 per cent. on \$60,000, amounts to \$9000 per year. At 18,000 lb. per day, 300 days per year, we shall produce 2410 tons of steel. This item therefore amounts to about \$3.74 per ton, bringing our total cost to \$77.59 per gross ton of steel in pots.

Estimating 60 per cent. good castings, we have \$77.59, less credit for .40 tons scrap at \$17.00 per ton (\$6.80) = \$70.79, cost of metal for .60 tons good castings; $\frac{70.79}{.60}$ = \$117.98 per gross ton of good castings.

SUMMARY

| | Per ton molten steel | Per ton good castings |
|--------------------------------|-------------------------|--------------------------|
| Basic open-hearth | \$28.86 | \$38.10 |
| Acid open-hearth | 31.57 | 43.28 |
| Acid electric-cold scrap | 37 - 47 | 53.11 |
| Side-blown Bessemer | 39.38 | 56.3 0 |
| Basic open-hearth and electric | 42.28 | |
| Bottom-blown Bessemer | 42.13 | 60.88 |
| Basic electric-cold scrap | 43.37 | 60.28 |
| Crucible—30-pot gas furnace | 77.59 | 117.98 |
| Crucible—coal holes | 129.27 | 204.10 |

Thus we see that the cost of the steel will run from about 1.6 cents to about 8.5 cents per pound of good castings. As casting prices range from about 4 cents on large tonnages of simple work

to 15 or 20 cents for what is generally considered average light work, it will be seen that the costly crucible process can be used only for the latter class of material, and that on the cheaper grades of castings generally produced in bulk, it is impossible to figure a profit with Bessemer or electric furnaces. The open-hearth foundries producing in bulk can shade the price the small amounts necessary to attract purchasers of heavy tonnages, and live.

The cost of steel in good castings is greatly affected by the percentage of good castings obtained from the steel in the ladle. Thus if we take the figure given above for basic electric steel in the ladle, (\$43.37), and assume that we obtain only 50 per cent. good castings, the rest being heads, gates and spillings, the cost of steel in good castings will be \$43.37, less credit for .5 ton scrap at \$18.00 per ton (\$9.00) = \$34.47, divided by .50 = \$68.94 per ton of good castings. Thus by decreasing the yield of good castings 10 per cent., we raise the cost of the metal in the castings \$8.66 per ton. We see from this how important it is to secure the maximum yield from the metal in the ladle.

A6. Tonnage.—The yearly tonnage to be produced affects rather the size and number of the furnaces to be installed than the choice of a process, as a little consideration of the foregoing discussion will show.

Production per week for the different processes is about as follows:

The crucible process, of course, can be used only for high-priced castings, and no possible increase of production can bring the costs down enough to enable this process to compete with the others on tonnage work. In the case of the open-hearth furnace, tonnage determines only size and number of furnaces. Though an increase of tonnage would somewhat reduce the cost of Bessemer metal, the small vessels can never be brought in direct competition with open-

hearth foundries, as the cost of steel does not decrease with the tonnage produced in the same proportion as in open-hearth shops. The Bessemer foundries therefore compete with the best of the acid openhearth steel makers, and with the users of the electric furnace and the crucible process.

The electric furnace at the present time is found most often in steel foundries making light and medium-light work, so that in most cases furnaces of not over 3 tons capacity are used; the 6-ton furnace has been installed in a number of shops with considerable success, but larger units are seldom seen in foundries. In practically all cases cold stock is charged to the electric furnace, and as melting cold stock in furnaces of much over 6 tons capacity leads to a number of difficulties that will be considered at length in a later chapter, there has been little incentive to the use of the larger units.

In the larger shops making ingots, electric furnaces often simply finish the refining of metal brought from the Bessemer converter or the open-hearth furnace. In these cases it has proved economical to install quite large furnaces, which produce steel considerably cheaper than the smaller units.

It may in some cases prove advantageous for an open-hearth foundry to install one or more small electric furnaces to refine open-hearth metal for the production of a limited tonnage of their lighter work. They would by this practice enjoy the advantage over the other makers of electric furnace steel of a considerably lower cost of steel in the ladle. Habit is compelling, however, and the tonnage shop that undertakes to go after a share of the small steel casting or specialty man's business must guard against the danger of failure that results from their own mental attitude; they must remember that the business has to be handled along different lines from the rest of their output, and that success in smaller steel castings depends very largely on the most minute and painstaking attention to details.

To feed a battery of electric furnaces with hot and partially refined metal from a tilting open-hearth furnace looks like an attractive method of increasing the tonnage and lowering the cost of electric steel, but there are a number of reasons why this practice has not so far been generally adopted. In the first place, it has been found impractical to run the open-hearth furnace continuously, as would be most desirable for this scheme, if the furnace is to be charged with cold pig and scrap. To take 3 tons of steel from a 15-ton open-hearth furnace and replace it with 3 tons of cold stock, has proved to be impossible, because the cold metal charged to the open-hearth

furnace so chills the whole bath that it cannot be made ready in time to take care of the next electric furnace. To take the whole heat from the open-hearth and refine it in one or more electric furnaces would be of no advantage, because the electric furnaces would have to lie idle so long between heats waiting for open-hearth metal, that the cost of the electric steel would be greatly increased.

The only way to succeed with such an installation, therefore, would be to charge hot metal into the open-hearth furnace, as the big makers of ingots do. This would mean melting the metal for the open-hearth furnace with a cupola, and would lead to so vexatious a problem in trying to keep the cupola, the open-hearth furnace and the electric furnaces "in step" that the advantage aimed at might very well be lost. It would, of course, be a difficult job to run a cupola at such a rate as to feed the open-hearth furnace at intervals; and as a last objection, the metal fed to the open-hearth would be too high in carbon to handle to the best advantage.

It should be perfectly feasible, however, to operate a plant consisting of cupola, side-blown Bessemer converters and electric furnaces; indeed such a plant is now in successful operation for the production of malleable iron. The advantages to be gained are cheap melting and partial refining in the cupola and converters, and low cost of final refining in the electric furnace, due to the shortening of the sojourn of the metal in the electric furnace. Off-setting these advantages are the greater loss of metal in the cupola and converters than in straight electric melting, the difficulty of keeping all the units "in step," and the higher cost of the raw material; for the electric furnace as a melting unit uses all scrap, while the cupola and converters would need a considerable proportion of pig, 40 per cent. at the lowest estimate. These disadvantages have so far outweighed the possible advantages in the minds of steel foundrymen, and the author knows of but one case where such an installation has been tried. Its failure in that one case is not, to the author's mind, proof positive that the method is not feasible, but any one contemplating such an installation should at least give it long and careful consideration.

The tonnage produced in electric-furnace plants has been increased mostly by speeding up the operation of the furnaces. By using charges that will melt down close to the composition in carbon desired, by using stock not too high in phosphorus and sulphur so that the refining period is shortened, by melting with all the power

¹A tilting open-hearth furnace has been used to feed a single electric furnace in this way, but the time between tops was quite long.

that the furnace will stand, and by speeding up recharging, the production of the basic furnace has in some cases been brought up to 8 heats in 24 hours.

The success of the acid furnace has been due very largely to the fact that more heats can be made in a day working acid than working basic, so that lower power, labor and other costs more than off-set the more expensive (low phosphorus) scrap required for acid work. Acid furnaces have averaged as high as 11 heats in 24 hours, making really good steel; and some plants using the acid furnace are turning out a heat almost every hour. These, however, are the shops that are doing the most to give the electric furnace a "black eye," as they are yielding to the temptation to entirely subordinate quality to tonnage, and in many cases turn out a very poor grade of steel. They tend to simply melt down a bath of scrap, make such additions as to bring the composition about right, and pour out their steel without making any real effort to do the careful refining that gives electric steel its real claim to being a superior product. In other words, by melting at the highest possible rate and doing practically no refining they have made the electric furnace a cheap melting unit; but there is no possible use in trying to hide the fact that the inevitable result of this practice is the production of cheap, inferior steel.

B. Raw Material and Fuel Most Available.—In the older ironmaking centers raw material for any of the processes is readily available, and hence is a secondary consideration in the choice of a process. Fuels may not be readily obtainable at a price to allow of the use of a process in a particular locality, especially where competing shops are already intrenched. This is especially the case with the electric furnace, whose fuel is electric power which it uses most uneconomically as compared with many of the other processes, in places where power must be generated from coal or oil. Exception has been taken to this statement by men who point out that the electric furnace uses the power delivered to it very economically indeed, as its thermal efficiency in terms of the percentage of the total power taken from the lines that is converted into heat is very This criticism, however, is entirely beside the point, as what we are interested in is the cost in dollars and cents of the heat we need for our steel; we have to consider the thermal efficiency, not simply of the electric furnace, but of the boilers, engine, generator, transmission lines, and transformers as well. When we add to this the labor cost involved in delivering current to the furnace and the interest and depreciation on the power plant and furnace as a whole, we readily see, when we look over the cost sheets for our metal, that the power bill for the electric furnace is much higher than the fuel bill for most, if not all, of the older processes. Where power is obtained from water, of course, or when it is purchased from a large power company, the cost is not so staggering, but even then it is plenty high enough.

The raw materials and fuels used in the processes are as follows:

| Process | Fuel | Raw material |
|-------------------|--|--|
| Crucible | Anthracite coal or coke. Producer gas (soft coal). Natural gas | Charcoal iron. |
| Bessemer | Coke (in the cupola) Power to drive blowing engines and compressors, obtained from coal, gas, oil, water, etc. | |
| Electric | Electric power, obtained from coal, natural gas, blast-furnace gas, coke- oven gas, oil, water. | Basic medium phosphorus scrap. open-hearth or converter metal. |
| Basic open hearth | Producer gas. Natural gas. Oil. (Coke-oven gas), etc. | |
| Acid open hearth | As above | Low phosphorus pig. Low phosphorus scrap. |

It is apparent that only in exceptional cases will the fuel and raw material available be the chief determining factor in the choice of a steel-making process. So many fuels can be used for each process that only in comparatively rare cases will only one be available, and even then this fuel will almost always be capable of use for several processes.

Electric power is frequently available at comparatively low figures

in districts where mineral fuels are costly, as for instance on the Pacific coast of the United States and Canada. In such cases the disadvantage under which the electric furnace labors in more favored districts may be entirely eliminated, and conditions point conclusively to the necessity of adopting this process to the exclusion of any other, on considerations of fuel alone. When it is remembered that for Bessemer practice the only fuel needed is the comparatively small amount of coke or coal used to melt pig iron in the cupola, or oil to melt it in a separate furnace, it will be seen that the mineral fuel must be very expensive and power very cheap, for this to be the case. The open-hearth furnace is so easily handled with oil, a very concentrated fuel, that it is rare for electric-furnace steel to be cheaper than open-hearth.

Again, where pig iron is extremely costly and scrap abundant and cheap, pig-iron consuming processes are at a disadvantage. This generally affects only the Bessemer process, leaving the choice among the other three open.

Even here, however, the problem can possibly be rendered soluble with the Bessemer process by melting a large proportion of scrap in the cupola, and obtaining the silicon necessary for the process from small amounts of ferrosilicon melted with the charge in the cupola, or melted separately and added before charging the mixture into the vessel. In this way, by paying freight on the silicon necessary for producing heat in the vessel, in very condensed form, the Bessemer process may be made applicable in the face of comparative scarcity of pig iron. The greatest difficulty that will be encountered in such a plant is the fact that it is very difficult to melt high proportions of steel scrap in a cupola without large losses from skulls in the troughs and ladles, as it is almost impossible to get the metal hot enough to prevent the formation of such skulls.

At least one process exists for adding carbon to steel scrap in the open-hearth furnace in the form of coke or some such substance, and thereby greatly reducing the amount of pig iron needed in the pig and scrap process. Thus, on our Pacific coast, advantage is taken of abundant oil fuel and cheap scrap to conduct the manufacture of open-hearth steel quite cheaply. With abundant oil and electric power and a large supply of scrap, the coast should in a very few years make itself independent of the East in steel castings.

C. Capital Available.—This is a question, of course, of first cost of plant, and is too often the cause of disaster with companies started with insufficient knowledge of the conditions to be faced. The proc-

esses, in order of their first cost of installation, the cheapest first, are as follows:

- 1. Crucible—coal holes.
- 2. Side-blown Bessemer.
- 3. Electric—(not counting power house).
- 4. Crucible—gas furnace.
- 5. Bottom-blown Bessemer—(including engines, boilers, etc.).
- 6. Open hearth.

From the point of view of the installation costs per ton of steel, however, as the figures already given show, this order is quite different. For convenience, the processes are again tabulated below, in order of installation cost per ton, as estimated above.

| Process | Interest and deprecia- tion per ton |
|--|--|
| Bottom-blown Bessemer | . \$.50 |
| Electric furnace, refining hot metal | 89 |
| Side-blown Bessemer | |
| Acid electric furnace, melting cold scrap | . 1.13 |
| Acid open-hearth | . I.24 |
| Basic open-hearth | . 1.40 |
| Basic electric furnace, melting cold scrap | . 2.09 |
| Crucible, coal holes | . 2.80 |
| Crucible, gas furnace | . 3.74 |

While there is much to be said in favor of starting a specialty or jobbing business on a small scale, especially when the men in charge are not thoroughly familiar with the work to be handled, it is easy to overdo this and start with so little capital that before the difficulties are all understood and overcome the management finds itself embarrassed from lack of funds. Thus if we mean to run a gas crucible steel furnace of 30 pots, and start our business with four or five coal holes, it is necessary to figure on a largely increased cost of metal at the start, and we may find it takes very much longer than we had expected to accumulate the earnings counted on to build the larger plant. The same is true, to a certain extent, of Bessemer and electric-furnace plants, indeed of any installation where the output is to be quite small at the start; since the expenses of supervision alone, to take one item, are necessarily much higher per ton for a very small than for a larger output. As a general rule, however, the safest plan is to start with the same process we intend to use ultimately, since each process has its peculiarities that must be mastered, and the experience gained with the small plant is useful with the larger one.

D. Competition.—A very few words must dispose of this heading, since it is too much a matter of plain business sense to need exposition.

Frequently, in a field already covered by tonnage foundries a good deal of business in small jobbing work can be picked up. To start a small plant to take care of such business is frequently profitable, since the work is generally high priced. It must be remembered, however, that it is "turned down" by the tonnage foundries because it is hard to make, hence it will not be plain sailing to turn out the castings. Moreover, the field will seldom be a large one and therefore the business can be handled only with a small tonnage equipment.

It is almost unnecessary to point out, after what has preceded, that to enter into competition with a firm well intrenched in a particular line, with a more expensive process than the competitor is using, is almost always suicidal, since extra prices are not readily paid for superior steel, if the established firm makes good sound castings. The new concern will get chiefly the "cats and dogs" turned down by the older shop, on which it is difficult to make a profit; and to try to get business by price concession is fatal, since the other fellow with his cheaper process can stand more than the newcomer in that line. Should one find that his shop has inadvertently got into such a situation, there are generally but three things to do—take the scrapings of the business, which if the controlling shop in the field carries on a tonnage business will often support a small concern; get some good specialties; or shut up shop.

E. Labor Available.—In districts where steel making is new, it is as easy to train men to one steel-making method as to another. Experienced men must be imported to start operations, but generally will not stay. The management must count, therefore, on training the local labor as rapidly as possible.

In the older steel-making districts, labor will frequently be available that understands one or two processes, but not the others. This is especially true of special designs of furnaces, as for instance the Krupp type of crucible furnace, which is now pretty dead in this country; trained labor for such an unfamiliar furnace cannot be obtained.

The same was true of the electric furnace only a few years ago, and even to-day really capable electric-furnace operators are by no means easy to find, and when found demand very high pay. It is probably safe to say, even now, that in many cases it is the best

policy to train one's own operators, using the experts of the concern installing the furnace as instructors. Great care must be exercised, moreover, in the choice of a furnace, as all too few concerns building electric furnaces have really capable operators in their employ. They may be able to state that they have a man of long experience, but it sometimes proves that his record has been mostly one of failure, and that he has taken to the wandering life of the man who installs and starts electric furnaces because he can no longer find employment as an operator.

Should the case for two processes be otherwise about even, a lack of men that understand one of them should at once put that process or special design of furnace out of the running. At times the difficulty of obtaining men to handle a process unfamiliar to the labor of the locality will even be found to convert a supposed superiority of a special design of furnace into a decided inferiority from the point of view of economical production of steel.

F. Intermittent or Steady Operation.—If the furnace can be kept going steadily, with no shut downs, any process may be used. Regenerative gas-fired furnaces (open hearth and crucible) cannot be shut down and started up again without great expense, and must be kept constantly hot. They are, therefore, best suited to double-turn operation at their maximum capacity. Operated single turn, the cost of steel will be greatly increased—and short shut downs will be costly, both because the furnace must be kept hot, and because to keep an expensive furnace idle runs rapidly into money.

The Bessemer converter is well suited to intermittent operation, and lends itself readily to very considerable fluctuations in the daily tonnage produced. The side-blown equipment being cheaper than the bottom-blown is less affected by shut downs.

Oil or coal burning crucible furnaces can be operated intermittently as easily as continuously, and their installation cost is so low that shut downs are relatively inexpensive.

Electric furnaces can be, and very often are, run only on day turn, and not always every day at that. When there is enough business to keep them running day and night, however, a considerable saving in cost of steel can be attained. It takes quite a little power, of course, to heat up a cold furnace, and the wear and tear on brickwork and the consumption and breakage of electrodes are considerably greater when the furnace is cooled down over night than when it is kept constantly hot. The interest and depreciation charges, of course, are higher for intermittent operation, and not as much steel

can be turned out per hour, because of the time lost in producing the first heat in a comparatively cold furnace. That shut downs of large (and hence costly) electric furnaces greatly increase the cost of steel, goes without saying.

SUMMARY

| Process | Quality | Flexi- bility | Suitability for small work | Cost of steel | Tonnage | Cost of instal- lation | Cost of installa- tion per ton |
|-------------------|---------|--------------------|----------------------------------|------------------|-----------|------------------------------|--------------------------------------|
| | I best | I most flexible | ı most suitable | I lowest | r highest | 1 lowest | 1 lowest |
| Crucible | 3 | I | ı | 7 | 6 | I | 7 |
| Basic electric | I | 2 | 2 | 6 | 5 | 3 | 6 |
| Acid electric | 2 | 2 | 2 | 3 | 4 | 3 | 3 |
| Acid open-hearth. | .4 | 4 | 5 | 2 | I | 5 | 4 |
| Basic open-hearth | 5 | 4 | 6 | ı | ı | 5 | 5 |
| Side blow Bes- | | | | | | | |
| semer | 6 | 3 | 3 | 4 | 3 | 2 | 2 |
| Bottom blown | | | | , | | | |
| Bessemer | 7 | 3 | 4 | 5 | 2 | 4 | ı |

Note.—Electric furnace refining basic open-hearth steel stands between No. 4 and No. 5 in cost of steel.

Note.—Gas-fired crucible furnace stands between 3 and 4 in cost of installation.

Note.—Electric furnace refining hot metal stands between No. 1 and No. 2 in cost of installation per ton.

CHAPTER III

THE CRUCIBLE PROCESS

The crucible process is the oldest of the steel-making methods now extensively used, and like practically all our modern processes is "indirect;" that is, steel is produced from some other product derived from the iron ore.

In very early times, both steel and iron were produced directly from the ore in open forges. These consisted essentially of an open basin or hearth in which very pure iron ore was heated in contact with a large excess of charcoal, the fires being blown with bellows or other primitive means of producing blast. Part of the ore was reduced by solid carbon, according to the usual formula,

$$FeO + C = Fe + CO$$
,

while a considerable amount of the FeO combined with the silica, lime, etc., of the gangue of the ore to form a fluid or pasty slag. The metal obtained from these forges was in the form of a coagulated mass of small particles, much like a puddled "ball," and was hammered to squeeze out the slag contained in the interstices, cut into pieces, heated to welding and rehammered, until a fairly pure and uniform bar was obtained. If the metal absorbed much carbon from the charcoal a steel of varying degree of hardness was obtained. As a rule, however, soft, carbonless iron was the product desired, and in order to convert this into steel it was heated in a bed of charcoal out of contact with the air until it was cemented or case hardened to the center.

Later, the soft iron used for cementation was produced from the new product, pig or cast iron, by melting a bar of pig iron with a charcoal fire in an open hearth, and allowing the slowly melting iron to trickle down through the air blast used. By the oxidizing action of the blast, the carbon, silicon, manganese, sulphur and phosphorus of the pig were eliminated and nearly pure iron was obtained in a sponge of coagulated particles and slag, which was worked up as before. By this method also either iron or steel could be obtained, but iron was the usual product. This is the principle of the Walloon

hearth and other like methods, by which Swedish charcoal iron is still made to-day.

The practice of cementation (or total case hardening, as it might be called), gradually developed as time went on, and as used in England to-day has made great strides in the furnaces employed. though in principle it is still the process of Tubal Cain. In the modern cementation furnaces, bars of soft iron are packed in long cast-iron boxes, each bar well bedded in pulverized charcoal, and two or more of the boxes are heated to full red heat in a conical topped, coal-fired furnace. After full heat is reached, it is maintained from 7 to 11 days, the progress of the absorption of carbon being watched by occasionally taking out a test bar through holes in the ends of the pots and when the desired degree of carburization has been reached the furnace is allowed to cool slowly. These "blister bars," as they are called, are broken when cool, the "temper" (or carbon content) estimated by the appearance of the fracture, and the bars are then hammered out into longer bars, a product known as "spring steel." Cut, piled, heated, welded, and hammered out into bars once or twice, the product is "single-shear" or "double-shear" steel. These steels are still extensively used in Sheffield for cutlery and many tools in which a steel edge is welded to an iron back.

About 1740, Huntsman, a watchmaker of Sheffield, dissatisfied with the far from uniform steel of his day, hit on the scheme of melting blister bar in clay pots or crucibles, and made the first crucible steel produced in England. Naturally this steel was a great improvement on "double-shear" and at once largely displaced all other steels for high-grade requirements. With characteristic conservatism, many Sheffield steel makers adhere to-day to Huntsman's methods, and for high-grade tool steels melt only blister bar of uniform temper, made from Swedish charcoal iron. Puddled iron, they maintain, will not make as fine a steel as the charcoal product, and is used in Sheffield only for poorer grades of steel.

It is hard to see the special virtue of spending two weeks soaking carbon into the iron in the solid state and then melting the product, when the two operations can be carried out simultaneously in a few hours. American crucible steel makers pack soft iron and charcoal (or washed metal), into pots, melt down the iron, which absorbs the charcoal or the carbon of the washed metal, and obtain a steel of the desired "temper" in 4 or 5 hours. Our precedent for this practice is found in the ancient steel-making industry of India,

Arabia, and other parts of the East, where for untold centuries steel has been made by re-melting in clay pots with wood or charcoal, iron produced directly from the ore in crude hearths similar to those already described. The pots are luted tightly, packed in a charcoal fire, and the contents melted as in American practice, the soft iron absorbing carbon from the wood or charcoal and becoming hard steel. By this process were produced the long famous blades of the Orient, of which we have all heard perhaps as much as we can stand, and which must be our justification for claiming that as good steel can be made by cementing and melting in one step as by the slow and costly methods so largely followed in Sheffield.

Puddled iron, invented by Cort about 1784, is the raw material chiefly used to-day in America for fine tool steel and when carefully made, with low sulphur and phosphorus we claim that it will produce a tool steel as good as any. Indeed, except that it is made with soft coal as fuel instead of charcoal, and out of a fluid bath not in contact with the fuel, instead of out of a succession of drops of fluid iron running through the air blast, it is hard to see wherein the metallurgical features of the process differ from those of the Walloon hearth. The claim is made that we do not so thoroughly work the slag out of our puddled bar as is the case in charcoal-iron practice, but since in re-melting in pots practically all slag floats out of the puddled iron, it is not easy to see the advantage of working the slag out of the solid iron by rolling and hammering. To suppose that the mechanical working imparts a "quality" or "body" to the iron that persists in the molten steel is too great a strain on our credulity to warrant serious consideration.

The crucible process has been used for so many years that tradition and superstition have gathered upon it like moss on an old wall, and this discussion is purposely made explicit and rather long in order to set forth the matter in what appears to the author its true light. Purchasers of castings so frequently used to state that after years of unsatisfactory experience with makers using other processes, they at last adopted crucible steel, and never had any more trouble, that it is as well to state clearly and at once that their relief from their troubles was chiefly due to the ease with which small and intricate castings of crucible steel are poured solid and free from blowholes, cracks and flaws, and to the pains that makers specializing in small castings habitually take with their product, and not to mysterious, cure-all virtues of the process itself.

Let us then consider for a moment the reasons for the excellence

of crucible steel, and the practice generally followed for the production of crucible steel castings, in order to see to what extent great excellence of steel is usually striven for and actually obtained. begin with, as crucible steel making is a pure melting process, no removal of phosphorus and sulphur is possible, and the content of these elements even increases slightly, because the loss of metallic iron reduces the total weight of the molten steel as compared to that of the iron charged; and in the case of sulphur, because some sulphur is absorbed from the coke, coal or gas used in melting. this reason low sulphur and phosphorus have to be obtained by the use of very pure raw material, either "low phosphorus" charcoal or puddled iron, or basic open-hearth scrap. If acid open-hearth or Bessemer scrap is used, the phosphorus and sulphur will be relatively high, and quite often by the indiscriminate use of plate-steel punchings whose origin is not known or whose analysis is not obtained, crucible steel is produced much higher in these objectionable elements than good practice should allow.

Excellence of the steel is further due to the fact that by melting in a closed pot with a cover either luted on, or soon sealed fast by the "running" of the pot and cover with the heat, the steel is protected from the oxidizing gases of the furnace, so that it does not oxidize and absorb its own oxides, nor absorb great quantities of harmful gases. There is, to be sure, air in the pot at the beginning, but the oxygen of this air is soon exhausted in the oxidation of charcoal in the pot, or of carbon, silicon and manganese of the iron, so that a neutral atmosphere is soon attained. This is, of course, lost when the cover is slid off to examine the steel, but the advantage from this source is well known and beyond dispute. Further, silicon, either absorbed from the clay of the pot by the reduction of silica, (SiO2), to silicon, by the carbon of the steel, or added as ferrosilicon. reduces much of the oxides present in the metal, and renders it much "quieter" when poured, either by freeing the steel from gas. or by increasing the capacity of the steel to hold the gas in solution. Thus in crucible melting, open-hearth scrap is considerably improved in quality, and if hard tool steel be the product, it is better than open-hearth steel as such.

Tool steel practice, however, which tries the steel very high, has demonstrated to the satisfaction of the steel makers and steel users that crucible steel made of open-hearth scrap, however low in phosphorus, sulphur and manganese, is not as good as crucible steel made from puddled iron. The English makers and users,

as already noted, carry this even further and will use only charcoal iron for the finest steels, and cement it before melting at that. As this superiority of tool steel made from wrought iron seems to be established beyond dispute, there must be some inherent excellence of wrought iron, not shown by the analysis for silicon, manganese, sulphur and phosphorus, which renders it superior as a raw material for crucible steel to open-hearth scrap of equivalent or superior analysis.

This excellence must be due to the method of manufacturing wrought iron, and to get an idea of what causes it, we must compare carefully the methods used for producing puddled iron and basic open-hearth steel of equal purity, as shown by the usual analysis.

In both the puddling furnace and the basic open-hearth steel furnace, a bath of molten iron high in carbon, silicon, manganese, sulphur and phosphorus is subjected to the action of iron ore in a highly basic slag, and of an oxidizing flame, in order to oxidize the impurities, and remove them either as gases (CO), or by sending them into the slag as oxides (SiO₂, MnO, etc.). The final product of both is nearly pure iron; but the temperature of the puddling furnace is so low that the metallic iron becomes solid as fast as it forms, a molecule or a tiny particle at a time, and by the gradual coagulation of these small masses a sponge of pasty iron particles is formed, whose interstices are filled with liquid slag; whereas the open-hearth furnace is maintained at so high a temperature near the end of the process that the metal is kept molten, poured into a ladle and thence into large ingots, where it cools relatively rapidly. The puddling process corresponds to the early stages of the basic open-hearth process, and it is owing to the lower temperatures that the slag, which is not very high in lime (CaO), and is very high in iron oxide (FeO), is able to retain phosphorus as phosphate of iron and largely eliminate this impurity. In basic open-hearth practice, though considerable phosphorus is held in the slag when the metal is first melted, much of this phosphorus is reduced again from the phosphate of iron by the action of iron and carbon at high temperature, and a very limey slag has to be made after the richly ferrous slag has performed its function of removing carbon, silicon and manganese, in order to hold phosphorus in solution and thus eliminate it from the steel by oxidation.

As in both processes the metal is exposed to highly oxidizing slag and gases, the difference must be chiefly in the lower temperature and the relatively slower and particle by particle solidification of the metal in the puddling furnace. It has always seemed probable that at the lower temperatures of the puddling furnace the metal absorbs less oxide and gases, especially since the very pure iron formed near the end of the process at once freezes, instead of being long exposed to oxidizing conditions when it has no silicon and manganese to protect it from oxidation, as in the case of the open-hearth furnace; and that the particle by particle solidification of wrought iron gives an opportunity for throwing dissolved gases and oxides out of solution that is largely denied to open-hearth steel frozen rapidly in large masses. To be sure, ferrosilicon and ferromanganese are added to remove a great part of the oxides, in open-hearth practice; yet they are often added in the ladle, and that their cleansing effect is far from complete is too well known to require argument.

From this discussion it is plain that crucible steel as such, is not necessarily a product of the highest excellence. If made of very pure materials properly melted, it is better than any other steel except the electric steel of to-day. If made of basic open-hearth scrap, even of very low phosphorus and sulphur, it is not a great deal better as steel than the scrap of which it was made. And if scrap is used indiscriminately without analysis, a very poor product may be the result. We have already set forth briefly in the introductory chapter our reasons for assigning superior quality to electric steel, and will dwell upon the matter more at length in the chapters devoted to the electric process. It is sufficient to point out here that the very moderate quality of crucible steel as generally made for castings is not at all as high as that of fine brands of tool steel, and cannot conceivably be called equal to the quality of well made electric furnace steel. It is just as well to state again, however, that for the great bulk of castings made by the crucible process, it is poor policy to use expensive puddled iron or charcoal iron in order to give the steel the fine quality of best tool steel. No casting is ever called upon to show the excellence of steel needed to make a tool stand up to longcontinued metal cutting, and castings properly made of crucible steel, using basic open-hearth steel scrap as raw material, will exhibit physical properties that show that they are amply able to endure the stresses to which they are to be subjected in service. More especially is this the case when the castings are annealed by simply a short heating and slow cooling to relieve stresses and somewhat improve the grain. We make our tools of the very finest steel in order that they may take the best possible temper when carefully hardened and tem-If we similarly heat treat our castings so as to give them the very highest strength and greatest toughness they can be made to possess, and find we need still finer steel than we can obtain from re-melted scrap, we naturally will turn to high-grade irons as raw materials, or to the electric furnace. But if we only roughly anneal our castings, and do not nearly bring out the best that is in them, there is no need of using such very costly raw materials.

Castings are generally desired of mild steel, .25 per cent. carbon being the average content for most purposes. Many makers of crucible steel castings have difficulty in producing mild steels, as will be explained presently when we consider the control of analysis, and produce a great deal of steel that for most uses is too hard and brittle for the best results. This is a disadvantage of the process that frequently gives the steel maker trouble.

The real advantages that make the crucible process well suited to the manufacture of castings, especially light and intricate castings, and alloy steels of certain kinds, have already been touched upon in the introductory chapter. They are:

First, the low cost of installation of the process for starting a small shop.

Second, the high temperature of the steel, and the fact that it can be kept hot in the furnace until needed. Moreover, when pouring small castings, the steel can be poured directly from the hot pot, which being a poor conductor of heat with a small exposed surface of metal keeps the steel hot and fluid very well indeed. This is the more true because the steel is not chilled by pouring in a stream into the pot, as in the case of filling small "shank" ladles from a larger ladle. In addition to this, the steel is very free from gas, is smooth running and if well "killed" or purposely made high in silicon, feeds down exceedingly well in the risers, "sets" quietly in the moulds, and produces very sound, clean castings. This advantage, coupled with cheapness of installation, are the two chief reasons for the use of the process, in spite of the high cost of the steel produced, for the manufacture of small, intricate castings.

An advantage of less weight in the majority of cases is the flexibility of the process, as already explained in the introductory chapter. Naturally, as each pot is a separate lot of steel, many different kinds of steel can be made at one heat in quantities to suit the requirements of the shop. The only limits to this are the number of pots melted, and the ability of the melter to attend to a number of different kinds of steel at once. We shall see further on that the difficulties of producing at will just the analysis desired, limit the possible number of sorts of steel that a melter can be expected to

produce successfully at one heat. There is not a very great demand moreover, for castings of special analysis, at least of steels that can be economically produced in crucibles, so that this flexibility is not often taken advantage of.

During the last 10 years, and especially since 1914, the crucible process has been used less and less in steel foundries. The competition of the users of the smaller sizes of electric furnace, who made a little better steel at a lower cost, was severely felt by the crucible shops, even before war conditions began to hamper the crucible The electric furnace has displaced the crucible process in the last 5 years, moreover, very much faster than it normally would have on account of the difficulty, even impossibility, of securing good "pots" during the war. Imports of graphite were greatly restricted, but the cutting off of the supply of clay for crucible making was the worst condition. Domestic supplies of suitable clay proved to be hard to find, and as a result the number of heats obtained from crucibles was greatly decreased, while the cost of the pots went up 500 or 600 per cent. These conditions made the lot of the crucible steel maker so hard that many foundries abandoned the process and turned to the use of the electric furnace, much sooner than would otherwise have been the case; and once the change was made, there was very little chance of the foundryman going back to the old process.

To-day, the average foundryman would hardly think of installing crucible furnaces in a new shop; Bessemer converters or electric furnaces would be used. In special cases it may be good policy to consider the crucible process, especially for a small output of steel castings, but the day of the installation of the process for a considerable tonnage of castings is past.

THE FURNACES

The furnaces used are the old-fashioned anthracite coal or coke hole, the oil-fired furnace, and the regenerative gas furnace of the pull-out (American), and side drawing (Krupp), types.

The coke hole is very little used in America, though coke is often used with anthracite in our practice in this country. The general style of hole used here for anthracite and in Sheffield for coke is about the same, and can be briefly described as a set of oblong melting holes lined with clay brick, with a common flue at the back into which a small flue leads from each melting hole. One stack provides

draught for a number of melting holes, and in America forced draught in the ash pits is universally used. The melting holes are sunk so that their top is at the working floor level, and there is a space provided in front of the ash-pit doors, so that the grate bars can be readily drawn from the holes for straightening, and the ashes easily shoveled out. This space is generally covered by a grating at the working floor level, which serves the double purpose of giving light for the men to see what they are doing when removing ashes, and allowing any spilled steel to drop through without getting under the

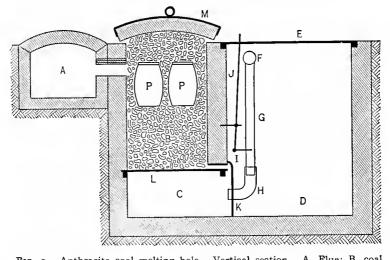


Fig. 1.—Anthracite coal melting hole. Vertical section. A, Flue; B, coal fire; C, ash pit; D, cellar; E, grating; F, G, H, blast pipe; K, ash pit door; I, J, blast damper; L, grate bars; M, cover; P, pots.

workmen's feet. The holes generally contain four pots, leaving room around the sides for fuel, and are deep enough to provide a bed of fuel from 18 in. to 2 ft. deep under the pots, when the tops of the latter are a few inches below the edge of the floor level. The covers of the holes are generally made in three sections so that they shall be light enough to handle easily.

The lay-out of such a shop is simple. The melting holes are generally lined up against one wall (or two walls) of the building, and the space in front of them covered with steel plates for piling plate scrap, heads, gates, and washed metal, for shoveling material and for packing pots. There should be room enough to pack at least one heat of pots comfortably, and have another heat packed ready for

charging. A storage place for new pots, used pots waiting to be recharged, etc., is needed, and a small room with bins and grocer's scales for storing and weighing the alloys, such as nickel and ferrochrome, needed for alloy steels, and the cracked ferromanganese and ferrosilicon weighed into manila envelopes and tied up to throw into the pots. It may in some cases be best to have the melting holes in a single or double row in the center of the shop, so that pots can be carried both up and down the pouring floor to get them rapidly out of the way. For ladle work, a small pit is provided, in which to set the ladle when it is being filled, and a small coal hole or a gas or oil burner is used for drying out the ladle. a bottom pour ladle is used, stoppers, nozzles, stopper rods and sleeve bricks for the same require a storage room, where stoppers and sleeves can be put on the rods and a few kept ready for use. Since plants of this sort are generally very small, the exact lay-out is largely governed by the sort of building used, and common sense must govern the placing of furnaces, etc., so that the material can be brought to them and the steel carried away with the minimum of confusion.

For a furnace of five holes, four pots in a hole, two heats per shift, the crew will be one melter, one pot puller, one moulder (as he is called in the tool steel shops), and perhaps one helper to shovel coal and ashes, help pack pots, and do odd jobs.

Sheffield Coke Holes.—In the Sheffield crucible steel industry, where fine steel is produced, clay pots are generally used and coke used for fuel. Clay pots are generally somewhat smaller than graphite pots, since they are weaker, and when first used are brought up to heat in the fires very slowly and then filled by means of a long iron funnel. In any case, whether first packed hot or not, they must be heated with extreme care and slowness, and after one heat has been melted are at once recharged hot and returned to the fires. If allowed to cool they will crack. Less steel is usually charged in the pot at each successive heat, owing to the deep cutting of the pot at the slag line. The coke holes often contain but two pots, but otherwise are much like the common anthracite melting hole.

The advantages of the coal- or coke-hole furnace, are its low first cost, and its suitability for intermittent operation. The chief disadvantage is the high cost of the steel, which is due to the low tonnage produced, the heavy wear on pots, which do not last as long as in gas-furnace practice, the high fuel consumption, and the high

labor cost. The pot pullers have more to do than on gas or oil furnaces, so that more of them are needed to handle a given tonnage, and the great amount of coal and ashes that have to be wheeled necessitates the employment of extra men. Another disadvantage is that in making very low-carbon steel, which is very hard to melt, the fires, especially on the second heat of the day, often give out completely before the steel is melted; and as it is not possible to raise the pots and shovel coal under them more than twice, it is sometimes impossible to melt this sort of steel on the second heat.

The Oil-fired Furnace.—The direct-fired oil furnaces are very simple in construction and comparatively inexpensive to build. In some furnaces of this type, there is a combustion chamber along-side the melting hole, in which the oil is burned, and the hot gases enter the melting chamber over a bridge. Many styles of oil burners are used. In some of them the oil overflows in a series of shallow pans and is carried into the furnace by a draught of air. In others

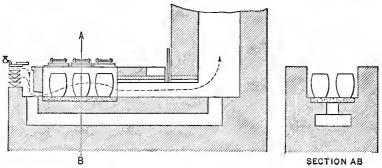


Fig. 2.—Milwaukee type oil furnace. Longitudinal and cross-sections. From "The Iron Trade Review."

steam or compressed air is used to atomize the oil. Whatever be the style of burner, the important thing is to secure complete combustion of the oil and proper distribution of the hot gases in the melting chamber.

As there are no coal and ashes to wheel, the crew of an oil furnace is smaller than that of a coal-hole plant of equal capacity. One melter and one pot puller, with one or two men to pack pots, can take care of a set of furnaces melting 40 pots a day.

The wear and tear of pots in oil melting is rather severe, yet the small crew and low fuel costs make these furnaces very economical when cheap fuel oil can be obtained. They have been extensively used in our middle West, where, however, the steady rise in the price of oil, and the increasing use of small electric furnaces have considerably altered conditions. The cost of installation of oil furnaces is higher than that of coal holes, but lower than that of regenerative

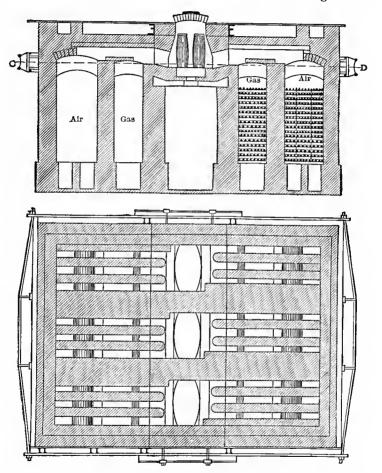


Fig. 3.—Siemens regenerative furnace. Vertical and horizontal sections.

From Stoughton, "The Metallurgy of Steel."

gas fired furnaces. Oil furnaces are well suited to intermittent operation, as they can be heated up very rapidly, and if built of good clay brick will withstand the heating and cooling involved. The melting is more rapid than in coal holes, and of course the pot pullers have to

handle the pots but twice, as they do not have to be raised as in coal hole melting; and as there is no bed of coal to hamper them, the men are able to grasp the pots more easily, and can be surer of avoiding weak places in old pots.

The Siemens Regenerative Furnace.—Regenerative gas fired furnaces, using either producer gas or natural gas, are of two types, only one of which (the melting hole design), is extensively used in America. These furnaces are built very much like gas fired soaking pits, with, of course, a smaller chamber, and work on the regenerative principle, which is too well understood to require extended description here. In regenerative furnaces the gas and the air necessary for the combustion enter and leave the furnace through fire brick chambers filled with brick checker work, which alternately are heated by the outgoing gases, and give their heat to the incoming air and gas. In natural gas furnaces only the air is preheated, and both sets of checkers are used for air, or only one set is provided.

The furnaces are sunk below the working floor, generally enough so that the floor of the furnace (level with top of the melting holes), is 2 or 3 ft. above the general level of the shop. The melting holes are covered with covers or "bungs" which are larger than those of coal holes, and therefore cannot easily be lifted off by hand with a hook, but are swung by a lifting bar suspended from an overhead trolley. Three covers to a hole is the usual and the most convenient arrangement. The holes may be lined with clay brick, especially if the furnace is to be run "single turn" and cooled off considerably at night to save fuel. In this case the resistance of clay brick to repeated heating and cooling is taken advantage of. Clay brick furnaces last on the average only 6 months, and then require extensive repairs. Generally, silica brick are used, and the furnace lasts 12 to 18 months. Six pots to the hole is the usual design.

The bottoms of the holes are covered 2 or 3 in. deep with coke breeze, to keep the pots from sticking to the bricks, and to soak up the runnings from the outside of the pots and any steel spilled from cracked pots. Openings about 5 in. wide are provided in the bottoms of the melting holes, which are closed with the bottoms of old pots. After a heat has been melted, these holes are opened, any slag or spilled steel poked through, the holes closed up again and a little more coke breeze shoveled in and leveled to make up for what has been poked out.

Reversing Valves.—The reversing valves used for regenerative crucible steel furnaces are the same as those provided for open-hearth

furnaces. For the air, of course, the plain "butterfly" flap valve is generally employed and gives satisfaction. The gas valve is frequently a butterfly, and should be water cooled. There is not the same objection to this style of valve for the gas in crucible furnaces that there is in open-hearth practice. Being smaller, the valves have less opportunity to warp; and as the waste gases going to the stack are somewhat cooler than in open-hearth practice, the warping is considerably less. The more complicated gas valves so valuable in open-hearth work are not often used for crucible steel furnaces.

Producers.—The scope of this book does not permit of a discussion of the types of producers for soft coal that can be used. progress has been made in the design of gas producers, with a view both to minimizing the hand poking needed in the old types and to increasing the economy of the producer. Good gas makers will keep almost any producer well poked down to prevent holes in the fire and consequent lean gas, but with the older designs the labor is severe. Many types should be regarded with a wary eye until their success in practice has been demonstrated, because some designers persist in giving great wind box area so distributed that after some months' operation the wind boxes get covered with chilled clinker, which spreads and chokes the whole fire and is very hard to poke out. Such producers give beautiful results on a short run, but kill the gas men after they begin to choke up. Water-sealed producers are thought by some furnace builders to be unsuitable for crucible furnaces, on the ground that the gas obtained from them wears out pots more rapidly than that made in dry-sealed producers.

In the lay-out of a shop using one of these furnaces, care should be taken to see that the melting shop and gas house can be thrown open on all sides in hot weather. The space for storing iron, scrap, etc., and packing pots, should if possible be on a level with the furnace floor so that pots can be "buggied" to and from the weighing space on a continuous floor. The pit between the regenerators into which the slag from the holes is poked should be easily accessible, at each end if possible; and at least should be so arranged that a good current of air can sweep through it. Otherwise it will get so hot in the pit as to make it impossible to clean out the accumulation of slag and coke without shutting down the furnace.

Ready access to all parts of the gas mains, valves, etc., is absolutely essential and manhole openings in the gas and air flues between valves and checker works will be found of great advantage

in facilitating cleaning out and minor repairs. Checkers, if possible, should be accessible front and back, as when the furnace has been long in operation the checker works become clogged with dust and have to be blown out with compressed air through openings in the end walls. This can best be done if access is to be had from both ends, but when the furnace is deeply sunk only one end is commonly accessible.

If producer gas is used the gas house should be provided with easy means of getting out the great quantity of ashes that have to be handled. If possible, the bottoms of the producers should not be in a pit of small dimensions, as it is in that case uncomfortably hot for the men to work in getting out ashes in hot weather. ashes have to be removed as they accumulate. With water-sealed producers this can be done at any time; with dry-sealed producers it is customary to do it on Wednesdays and on Saturdays during the general clean-out. On Wednesdays, during the cleaning out of ashes, it is well to open the gas mains and burn out the accumulated soot as much as possible. Saturday afternoon, after the last heat is drawn, is the time for general clean-out and any repairs to brickwork that are needed. The top holes of the producers are opened and the gas allowed to burn, all doors in gas mains are opened, the soot is burned out of the mains and valves as completely as possible, and the rest dragged out with wooden scoops. This is the time to give the neighborhood a wide berth unless your duty compels you to boss the job; and in hot weather, all hands especially welcome the feature of an open-sided shop during cleaning out. The floating soot is bad enough even in such a shop.

Starting the Regenerative Furnace.—In starting a new producer, it should be filled up with ashes well over the tuyères or wind boxes, and the fire kindled upon the ashes, coal being dumped in as soon as the wood fire is burning well, and the blast used only as needed. No gas should be turned into the furnace until good gas is being made.

While the producers are being started, the furnace is dried out with a coke or anthracite fire in each hole. The stack damper is raised, the valves are set on center, the covers of the holes are opened a little, and the draught is taken in the top of the holes, through all checkers, through the flues to the valves, and out the stack flue and stack. In a couple of days the furnace will be hot enough to light, when the valves are thrown over, the covers of the holes swung further apart, and gas carefully admitted, then a little air. Care

should be exercised in not turning on gas too rapidly, or too much air, or air first, as a strong "kick" frequently occurs in any case, and a bad one may damage the furnace. After an hour or so the valves are thrown over; it is as well on the first two or three reversals to shut off gas and air before throwing the valves, to avoid strong "kicks." After a few reversals the valves can be thrown over without this precaution and when the furnace has become hot the reversal should take place without any kick at all. As the furnace heats up, the reversals are made at shorter intervals, and when it has reached working heat should be reversed every 15 or 20 minutes.

The crew of a 30-pot gas furnace consists of one melter, two pot pullers, one or two moulders, and when producer gas is used, two gas men per shift. The moulders help pack pots, wheel them to and from the holes and do odd jobs, and one of the gas men is used to swing the covers of the melting holes when pots are being charged or drawn. If the pots are poured one at a time, the moulders wheel them to the edge of the furnace and wipe the slag from them; the pouring gang from the foundry then carry them from the furnace. In pouring into a ladle, speed of drawing is essential; in this work the melter pours the pots as fast as they are brought to him, the moulders run them to him with buggies, and the pot pullers draw as fast as they can, one man pulling six pots while the other gets his breath and wets down his clothes again. In this way 30 pots can be drawn and dumped into the ladle in 4 or 5 minutes. Three heats should be melted per shift.

Advantages of the Gas Furnace.—The gas furnace has the advantage of clean and easy handling of pots already mentioned in the description of oil furnaces, and like the oil furnace, gives the melter almost complete control of the temperature at all times. The stack damper, air supply and gas supply valves enable the heat of the furnace to be closely controlled, and in addition by opening the covers of the hotter holes a little, they can be held back when melting too fast and getting ahead of the others. There is, however, no means of controlling the admission of gas and air in each hole, as the whole furnace is handled with one set of valves. advantages of this type of furnace are low fuel cost, low labor cost, and speed of melting. The only disadvantages are high cost of installation, and the fact that single turn operation is far from economical. If no steel is melted at night, one or two men have to be kept on to make gas and reverse the furnace every 20 minutes or hour, and though the furnace is not kept up to full heat, fuel is burned and labor paid without return, resulting in a considerable increase of cost, to say nothing of the overhead cost of keeping an expensive furnace idle half the time. The necessity for the employment of pot pullers has been referred to as a disadvantage of these furnaces which does not apply to the Krupp type, in which the pots are lifted out with long suspended tongs and regular pot pullers are not employed. However, melters and pot pullers in this country are clannish and much inclined to stick together, so that a shop using the Krupp furnace would be certain to find it difficult to secure competent melters. This offsets the advantage of pulling pots with cheaper labor.

The Krupp Furnace.—The Krupp furnace, which has been used in one or two shops in this country, is built a great deal like a small

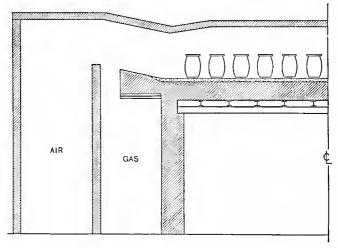


Fig. 4.—Regenerative crucible furnace—Krupp type. Half longitudinal section.

open-hearth furnace. The pots are set in the furnace and pulled out through doors in the side by means of a long pair of horizontal tongs suspended from a trolley. The bottom of the furnace is nearly flat and commonly slopes toward a tap hole in the back through which slag and spilled steel are run out. The bottom is made of silica sand fused on in thin layers. Except that the bottom is flat and the port construction not at all the same, the furnace resembles in its general appearance a small open-hearth furnace. Its advantages are those of the other type of regenerative furnace. Its disad-

vantages are, that to see into the pots it is necessary to go up on the top of the furnace and peep through six or more small holes in the furnace roof; that spilled steel exposed to the gases of the furnace soon is "puddled," or in other words the carbon is oxidized out of it and carbonless iron formed, which will not run out the tap hole and has to be pried off the bottom, often taking part of the bottom with it; and that as the furnace is a "freak" in this country, men have to learn to run it instead of going right in and handling the The disadvantage that gave the author furnace well from the start. the greatest trouble when handling one of these furnaces was the spilled steel, which worked its way into the bottom, because the scale formed on it cut out the sand. The "skull" that accumulated often had to be burned out of the furnace from the tap-hole side with an air blast. In Germany, clay pots are used in these furnaces, the pots are never worked to the limit, and few break; but in America we work so constantly to get the last possible pound of steel out of a pot, that some break in almost every heat of old pots, and trouble results.

The Pots.—Pots are made either of clay, or of graphite and clay in about equal parts (the common graphite pots). The disadvantages of clay pots already mentioned are so pronounced that graphite pots with a clay lining about \(\frac{1}{4} \) in. thick are often used in place of them. These clay-lined pots do not have to be handled so carefully as those made entirely of clay. The reason why in some instances clay, or at least clay-lined, pots are indispensable in good foundry practice will be taken up later, when the control of analysis of crucible steel is considered. Clay pots seldom make over three heats, while graphite pots will sometimes make five or six in foundry practice. During the war, as already explained, the quality of pots fell off to such an extent that one felt lucky to get even one or two heats out of them without trouble. Graphite pots should be stored in a dry, and, if possible, warm place; should be dried out on a furnace top for a week or two before using, and when new should be heated around the melting holes before charging. When hot they will stand pretty rough usuage, and can be dropped from the furnace to the ground without breaking if let fall on their butt ends and rolled out In the rapid work of ladle pouring this generally has to be done. When cool, the pots are examined for thickness, and tested for cracks by tapping them with an iron rod. If all right they ring clear like a bell, if cracked they will not ring. If not cracked, and thick enough for further use they are taken to the packing floor for repacking. If they are cracked or worn out, the bottoms are cut off with a coarse saw, or the sides smashed away with a heavy blunt knife, and the thick bottom parts are used as covers for the pots.

Melting.—In coal holes, a good grade of "broken" anthracite coal is used for fuel, and about two heats can be melted per shift. When operating single turn, the fires are lighted early in the morning, and blown up until a good bed of glowing coals about a foot deep is provided. About 6 in. to 1 ft. of coal is then dumped in, and the pots set directly upon it. The pot puller sets each pot firmly into the bed of coal, and when all are in, the moulder fills up around them with coal, heaps it up about 8 in. over the tops of the holes, and puts on the covers. Care must be taken to keep the coal out of the pots, as the steel absorbs carbon and sulphur from it very rapidly. The melter then puts the forced draft on the fires and urges them. At first the fires should be blown rather gently, especially if new pots are being used, because if the pots are heated up too rapidly they will spall and crack.

The melter should not be allowed to rush his fires up to full heat at the very first, because the sides of the pots are a poor conductor of heat, and through them must be driven the total amount of heat necessary to bring the steel up to the melting point, and to melt it; and if the outsides of the pots are brought to white heat very rapidly, leaving the contents of the pots comparatively cold, the heating up of the steel is not greatly hastened, but the pots themselves are exposed to extreme temperatures much longer than necessary, and are melted and slagged away on the outside very rapidly.

After about 2 hours the fires and pots settle down in the holes, so that it is necessary to get more fuel in under the pots to keep them off the grate bars. The pot puller goes over the holes and raises each pot with his tongs, while the melter pokes the coal well in under them, and the moulder shovels in more fuel. For this purpose coal is generally used, though sometimes part or all of this extra fuel is coke, which burns up more rapidly and makes a more intense heat than coal but does not last so long. The fuel is bedded up about level with the tops of the pots and well poked in, the covers replaced on the holes, and the fires urged again. In from 1 to 3 hours more the steel will be melted.

The melter examines the pots by separating the covers of the holes a little with his poker, and sliding the lid off each pot in suc-

cession. If the steel is melted, he thrusts his poker to the bottom of the pot to ascertain the condition of the metal. When first melted the steel boils quite freely and is "cold," so that a layer of it sticks to the poker. The boiling gradually subsides, and when the steel has been held in the pots a sufficient length of time, the bubbles in the slag will be large, and burst lazily, forming the socalled "cats-eyes." When the steel is hot enough not to stick to the poker and is quiet, the melter tosses in the additions of ferrosilicon and ferromanganese in envelopes and after a few minutes the covers are removed from the holes and the pots drawn. The adhering coal is rapidly scraped from the sides of the pots, the lids removed, and the slag wiped off. This is done with a slag "mop," or ball of slag formed on the end of an iron rod, with a ring at the end to start the formation of the slag ball. By wiping around the pot with the mop, at the same time turning it in the hand, a layer of sticky slag is caught on it which is spun off by rapidly whirling the mop along the floor. In a very few moments the steel is wiped clean and is ready to be carried to the moulds. In pouring into the moulds any remaining slag or scum is held back with a skimmer, or "flux-stick."

When a ladle is used the melter rapidly dumps the pots into it without wiping off the slag, which forms a blanket over the steel and protects it from chilling. For this practice, the more slag the better.

In oil and gas furnaces the melting is carried out very much in the same manner as in coal holes, with the exception that the pots naturally do not have to be raised to put extra fuel under them, and that the melting is more rapid. When using the Krupp type of furnace, it is not possible to make final additions to the pots in the furnace, and ferrosilicon, etc., must either be charged with the other stock, or in the ladle if one is used.

In both oil and gas furnace melting, the remarks already made on the dangers of too rapid firing at the first apply with even greater force than for coal hole melting, as it is easier to obtain extreme temperatures soon after the pots are charged. With bad handling, too hot a flame at first, and too much air admitted with the gas or oil, new pots are sometimes completely used up in one heat; properly handled, pots should average in a gas furnace three or four heats in steel foundry practice. When high carbon steel is melted, they will last much longer.

THE RAW MATERIAL

The raw materials used in crucible steel foundries are:

Puddled iron.

Charcoal iron.

(Basic) open-hearth scrap, especially boiler plate punchings.

Washed metal.

Charcoal.

Ferrosilicon, generally 50 per cent.

Ferromanganese, generally 80 per cent.

Nickel, ferrochrome, metallic chrome, tungsten, molybdenum, etc.

(Head, gates and scrap castings.)

Specifications for the first four on the list should be carefully drawn so that the phosphorus and sulphur of the steel shall be within the limits desired. If castings are made to recognized specifications, a top limit of .08 per cent. of phosphorus and sulphur will be demanded for those on which no tensile tests and bend tests are made, and .05 or .06 per cent. where these tests are called for. In any case the phosphorus and sulphur for good castings should never be allowed to exceed the upper limit of .08 per cent.

In order to keep within these limits, the phosphorus and sulphur in the raw material have to be somewhat lower than this, since there is some loss of iron in melting, and the impurities therefore increase; some phosphorus comes from the ferromanganese, etc., used; and a little sulphur is picked up from the flame; and as it is necessary for good economy to re-melt the scrap made in the shop the increase works up to a limit.¹

Assuming that we melt in each pot 30 lb. of our own scrap, $69\frac{1}{2}$ lb. of iron (or boiler plate punchings) and washed metal together, and $\frac{1}{2}$ lb. of ferromanganese; that the loss in melting is 2 per cent.; that the ferromanganese contains .3 per cent. phosphorus; and that the maximum percentage of phosphorus we can allow in our steel is .05 per cent.; we can figure the maximum allowable percentage of phosphorus in our iron and washed metal as follows:

Let x equal this allowable percentage; then,

$$.05 = \frac{69.5(x) + 30(.05) + .5(.3)}{98}$$

whence, x equals .046 per cent.

Practically, it is better to keep below this figure. Swedish (char¹ See Chapter XII.

coal) iron and puddled iron can be bought at .012 per cent. or .02 per cent. in both phosphorus and sulphur, though .03 per cent. is a safe limit, and cheaper material can be bought at this figure. Washed metal can easily be purchased below .02 per cent. or .025 per cent. in these impurities, and had better be kept to this figure, since the expense is generally not greater than for material analyzing .03 per cent. Open-hearth scrap can be bought below .03 per cent.. and .04 per cent. is the highest that can safely be allowed, since the determination of the exact analysis of an entire lot is impossible, and some of each shipment may run higher than the average. Ferrosilicon and the other alloys, except ferromanganese, are commonly low enough in phosphorus and sulphur not to need attention. Ferromanganese frequently runs high in phosphorus, and the German product especially should be taken only on guarantee of its phosphorus content, as it runs higher in that impurity than English ferro. That made in this country has been the lowest in phosphorus. The average contents in other constituents can be taken about as shown in the table on page 66: those to be specified are in parentheses.

THE CONTROL OF ANALYSIS

In figuring the analysis of crucible steel, nearly everything is fairly easy to estimate, except the carbon. While part of the carbon is contained in the alloys added and the iron and washed metal (or charcoal) melted, a considerable amount is absorbed from the graphite of the pots. This amount will vary from .15 or .20 per cent. to .40 per cent. according to the following factors:

- 1. Scrap and iron high in manganese absorb more than those low in manganese.
- 2. Heats hard to melt (puddled iron or steel scrap and a little washed metal) absorb more than heats easy to melt (all our own scrap for instance).
 - 3. New pots contribute more than old.
 - 4. Heats held long after melting pick up carbon rapidly.

Just how much will be gained can only be learned by experience and experiment, and mixtures frequently have to be changed to keep analysis correct. As a gas furnace grows old it melts more slowly, and commonly the steel then absorbs more carbon. Pots melted in gas and oil furnaces do not pick up as much carbon as those melted in coal holes.

For melting soft steels, below about .25 per cent. carbon (or even higher in the case of alloy steels), a clay or clay-lined pot must be

| Material | C, per cent. | Si, per cent. | Mn, per cent. | S, per cent. | P, per cent. | Ni, per cent. | Cr, per cent. | W, per cent. | Mo, per cent. | Slag, per cent. |
|-------------------------|--------------------|---------------------|---------------------|--------------------|--------------------|---------------------|---------------------|--------------------|---------------------|-----------------------|
| | 01. | | 01. | (.012) | (.012) | | | | : | I-3 |
| basic open-hearth scrap | . 10 to | .08 to | .40 to . | (.03) | | | | | | |
| Washed metal | about 3 | 01. | OI. | (.025) | (.025) | | : | : | : | : |
| Ferrosilicon | 2.00 | (o1) | 3-4 | .03 | | : | : | | | : |
| Ferrosilicon | . o8 to 2 | (50) | : | .03 | .03 | : | : | : | : | : |
| Ferromanganese | 6.00 to | . 20 to | (80) | .03 | .30 | : | : | | | |
| | 7.00 | .50 | · | | | | | | | |
| Silicomanganese | I.00 | (20 to | (50 to | .03 | .05 | : | : | : | | : |
| Ferrochrome | (.6 to | 25) .I to | (0/ | 80. | 70. | | (70) | | | |
| | (8 | 0.1 | | | • | | ; | | | |
| Tungsten ¹ | (.o3 to | os to | : | or to.) | 800.) | | | (8606) | | |
| Molybdenum ¹ | .5) (up to | .2 to .4 | | .3) (.03 to | to .02) | | | | (80-00) | |
| Nickel | .5. | н. | : | .3) | to .02) | (86) | | | | |

¹ Of late years, ferrotungsten and ferromolybdenum are displacing the pure metals for steel making.

used, to prevent the absorption of carbon. In fact, there is a loss of from .15 per cent. to .40 per cent. carbon in clay pot melting, which varies as follows:

- 1. Heats hard to melt lose more than those easy to melt.
- 2. Heats held long after melting lose more.

The clay lining of these pots is sometimes cut through to the graphite by the slag, especially if the slag formed is high in MnO, and then the steel absorbs carbon and the analysis obtained is not what was expected. This seldom happens on a new pot, or a pot used the second time on steel easy to melt. After the second heat there is not often enough of the lining left to use the pot again for making low carbon steel, unless the steel that has been made was very easy to melt. When the lining is cut through, the pot can be used for high-carbon steel, allowing for a smaller absorption of carbon than for a regular graphite pot as long as any considerable amount of lining remains.

Silicon and manganese, when added in the form of 50 per cent. ferrosilicon and 80 per cent. ferromanganese, frequently come out lower than expected, if the alloys are charged in the pots with the iron and scrap. They are best added cracked fine and weighed into manilla envelopes, which are tied up with twine and tossed into the pots about 5 minutes before drawing. This procedure greatly shortens the time formerly allowed for "killing" or "dead melting" (allowing the steel to absorb silicon from the clay of the pot), which was once from $\frac{1}{2}$ to $1\frac{3}{4}$ hours. Dead melting is now seldom practiced, and the steel is not held long in the pots after it has become quiet.

The alloys used, such as chrome, tungsten, molybdenum and nickel, are charged with the iron or scrap. The metallic nickel on the market is nearly pure, and should generally be figured as containing 98 per cent. nickel. Molybdenum and tungsten, when used in the metallic (powdered) form can be figured, if of about 98 per cent. purity, as containing 95 per cent. metal. When the ferro alloys are used, allowance must be made for some loss. Ferrochromes generally contain 70 per cent. chrome, and can be figured as containing 65 per cent. Carbon, silicon and manganese in all materials charged, if of sufficient quantity to affect the results, should be carefully figured in, not omitting to figure the carbon in the ferrosilicon and ferromanganese used as final additions.

If charcoal is used as the source of carbon, the amount it will contribute is variable, according to the difficulty of melting the steel.

Easily melted charges absorb most carbon from charcoal. It is a safe rule to figure the charcoal as containing 85 per cent. carbon on heats of all high carbon scrap, and as low as 70 per cent. or even 65 per cent. in all puddled iron (or plate scrap) heats of low carbon content.

Without close cooperation between steel maker (figure man), and melter, the control of analysis, especially of carbon, is impossible. The pots never melt at a uniform rate, and, as already stated, the rate of melting affects the amount of carbon absorbed from the graphite very greatly. The pots nearest the ports in a Krupp furnace will melt first; in regenerative hole furnaces, generally the hole nearest the gas valves leads; in coal holes, the fires draw better in one hole than in another; and in any hole the middle pots generally lead the end pots. Hence some pots will melt more slowly than others, and since it is commonly not practicable to draw the pots as they melt, some will be held fluid after melting longer than others. In hole furnaces it is, of course, possible to draw all the pots in one hole as soon as they are melted, and even in Krupp furnaces this can be done to some extent. The furnace crews, however, naturally do not relish being kept about in their wet rags very long, and it is seldom good policy to start drawing until all pots are far enough along so that they can all be pulled without much delay. Another reason why intermittent drawing is bad practice is that in drawing the furnace cools off considerably, which retards the melting of the laggards; and that the melter being busy bossing the job of drawing cannot give his attention as he should to the steel being melted.

In hole furnaces, the melter has an opportunity to slow down the holes that are melting too fast, either by shutting off the forced draught on coal holes, or by opening the covers on gas holes. On oil furnaces, the flame can be cut down for the same purpose. The good melter watches his steel carefully, and by noting the rapidity of melting, the boiling of the steel, the depth of cutting at the slag line, etc., can gauge with surprising accuracy the progress of absorption of carbon, and draw his heat at such a point that the majority of the pots will give steel close to the desired composition. But inevitably there will be variations, frequently quite wide, between the analyses of individual pots.

The same is true, with a reverse sign, of melting in clay pots or clay-lined pots; indeed, it is seldom possible in clay-lined pot practice to obtain the same nicety of control that is characteristic of higher carbon steel making in graphite pots.

THE USE OF LADLES TO PROMOTE UNIFORMITY

Here we see in greater detail one of the disadvantages already mentioned of crucible steel making, the auto flexibility, so to speak, of the process, which often causes wide variations in the analysis. and hence the physical properties, of the castings produced. tool steel practice, mixing the steel in large ingots by pouring a number of pots through a common funnel, one after the other, or pouring three pots into two and then into the ingot mould, has proved but a partial corrective of this evil. The large ingots, say of 8 or 10 pots, so poured, have proved very far from uniform along their length and cross-section, as there is not time in the moulds for thorough mixing of the steel from the different pots; and the same will commonly be true of steel castings poured by dumping pots successively into a casting. The corrective for this trouble is mixing the steel in a ladle. Necessarily, in ladle pouring, all the pots have to be melted before any are drawn, but a good melter will often draw some pretty cold ones, balancing them with hot steel from other pots, when he feels that the average of the heat has reached the analysis desired, and will turn out steel of surprising uniformity from heat after heat. It is never, however, possible to guarantee the same exactness of composition that is possible for instance in acid open-hearth practice, and a leeway of 10 "points" at the very least, as .30 per cent. to .40 per cent. when .35 per cent. is desired, is the closest control of carbon that can be expected for a number of heats.

There is one disadvantage in ladle pouring, however, in that it sacrifices some of the heat of the metal. Thus, although the steel can be got quickly into the ladle and is protected from cooling by the blanket of slag (which is purposely made abundant for ladle work by putting about a cupful of brick dust in each pot), yet it naturally does not keep as hot as if left in the furnace till needed. Hence, in pouring very small work, even if the steel after mixing in the ladle is promptly distributed about the shop in shank ladles, more difficulty will be experienced than in pouring directly from the pots. It is, therefore, a question of judgment how far to go toward securing uniformity of composition by ladle mixing at the sacrifice of the high temperature so necessary for running small and intricate work. We are more or less between the devil and the deep sea: to leave pots too long in the furnace because the pouring gang is small and few pots can be handled at a time, increases heterogen-

eity; to draw them all and mix the steel in a ladle often means lost castings or metal poured into scrap because it has grown too cold to run thin sections. Again, a large pouring gang can handle the pots rapidly from the furnace, if they are poured direct, or can handle a number of small shanks at once if the steel is mixed in the ladle, but a large gang costs money. Moreover, to pour small castings over the lip of a large ladle, or through a nozzle, is out of the question, as the stream is too large and there is no chance to pour the castings at the rather gentle rate often necessary. Perhaps the ideal condition would be to have enough heavy work on hand to take the cooler metal from the ladle after about half the heat had been poured into light castings by means of small shanks filled from the ladle. The whole subject is a nice question of judgment, and is discussed at some length here, because though similar conditions exist in pouring steel in the other steel-making processes, they do not to any extent involve the non-uniformity of composition so hard to deal with in crucible steel making.

As the troubles of the melter in securing steel of proper composition from a whole heat are many and various, it is clear that to require him to produce more than three or four different kinds of steel in one heat so multiplies the chances of error as to practically insure a good deal of steel of faulty composition.

Packing Pots.—In packing pots, a little iron (or plate scrap) is first laid on the bottom, then the charcoal, nickel, ferrochrome, etc. (brought in weigh pans from the "psychic" room), then the rest of the iron and scrap. Washed metal is frequently charged in good-sized lumps on the very top, leaving the cover raised some inches—in the opinion of the author this is poor practice, as some of this metal is apt to run down the outside of the pot and cut it badly. Pots should never be charged more than level full, and though putting the washed metal on the top facilitates melting because the high carbon washed metal melts first and in running over the iron and scrap helps carburize and melt them, yet in the process some carbon is inevitably oxidized which would not be lost if the washed metal were covered with a good layer of iron or scrap.

Examples of Charges.—In figuring crucible steel charges, the author has found the most convenient method is to work entirely in ounces, a 90-lb. pot containing, of course, 1440 oz. This is a comparatively easy number to divide by, especially since one soon learns its multiples; and the larger figures so involved will be found easier to handle than fractions or decimals of a pound, which must

afterward be reduced to the nearest ounces in weighing out the charges. For the benefit of the steel maker who may be interested in this method of figuring, and for the elucidation of some of the remarks already made on gains and losses in melting, the following typical cases are appended.

The lime in these examples is to help form a slag, by uniting with the iron oxide of the rust and scale on the iron and some silica from the clay of the pot. The black oxide of manganese is added with the idea that manganese will be reduced from it by the carbon of the steel, resulting in the better deoxidation of the metal. Its use in conjunction with ferromanganese is of doubtful value.

The "Base" is the estimated weight of the steel, from which the percentages of carbon, silicon, etc., are calculated.

(1) HIGH-SPEED TOOL STEEL-GRAPHITE POTS-COAL HOLES ½ oz. 80 per cent, ferromanganese..... 1 oz. = 70 per cent. ferrochrome...... 4 lb. 8 oz. = 72 oz. Tungsten..... 8 lb. 12 oz. = 140 oz. Molybdenum..... 2 lb. 14 oz. = 46 oz. Ferromanganese in package..... 1460 oz. = 1440 oz. CARBON Ferromanganese..... 2 oz. at . o6 = .12 oz. Tungsten...... 140 oz. at . o1 = 1.40 oz. Molybdenum..... 46 oz. at .o15 = .69 oz. Iron...... 1200 oz. at .001 = 1.20 oz. 8.45 oz. =.59 per cent. CHROME 72 oz. at .65 = 46.8 oz. = 3.25 per cent. TUNGSTEN 140 oz. at .95 = 133 oz. = 9.24 per cent. Tungsten.... MOLYBDENUM 46 oz. at .95 = 43.7 oz. = 3.03 per cent.Molybdenum..... MANGANESE 2 oz. at .75 = 1.50 oz. Ferromanganese..... Iron..... 1,200 oz. at .001 = 1.20 oz. 2.70 oz. = .19 per cent.

| ANALYSIS OF ST | EEL |
|----------------|-----|
|----------------|-----|

| | | | | | | S | |
|-----|--------|------|------|------|-----|------|------|
| .87 | 3 · 39 | 9.31 | 2.92 | . 19 | .02 | .014 | . 12 |

(2) High-speed Steel—Clay-lined Pots—Coal Holes Charge

| Lime | $\frac{1}{2}$ oz. |
|-----------------------------|--|
| 80 per cent. ferromanganese | $3^{\frac{1}{2}}$ oz. = 3.5 oz. |
| 70 per cent. ferrochrome | 4 lb. 8 oz. = 72.0 oz. |
| Tungsten | 8 lb. $12 \text{ oz.} = 140.0 \text{ oz.}$ |
| Molybdenum | 2 lb. 14 oz. = 46.0 oz. |
| Wrought iron | 74 lb. $= 1184.0 \text{ oz.}$ |
| Ferromanganese in package | = 2.5 oz. |
| - | |
| | |

1448.0 oz.

Base..... = 1440.0 oz.

CARBON

| Ferromanganese | 6 oz. at .06 | = .36 | oz. |
|----------------|------------------|----------|-----|
| Ferrochrome | 72 oz. at .07 | = 5.04 | oz. |
| Tungsten | 140 oz. at .01 | = r . 40 | oz. |
| Molybdenum | 46 oz. at .015 | = .69 | oz. |
| Iron | 1184 oz. at .oor | = r.184 | oz. |

8.674 oz. = .60 per cent.

Chrome, tungsten and molybdenum same as (r).

MANGANESE

5.684 oz. = .39 per cent.

ANALYSIS OF STEEL

C Cr W Mo Mn P S Si .40 3.21 9.01 3.25 .10 .013 .014 .09

(3) High Per Cent. Nickel Steel—New Clay-lined Pot—Coal Holes Melting Time 4 hr. 51 m.

CHARGE

| Lime | ₹ OZ. | | |
|-----------------------------|-------------|--------------|--|
| Black oxide of manganese | 2 OZ. | | |
| Charcoal | 4.5 oz. | = 4.5 oz. | |
| Nickel | 17 lb.4 oz. | = 276.0 oz. | |
| 80 per cent. ferromanganese | I.5 OZ. | = 1.5 oz. | |
| Iron | 73 lb. | = rr68.o oz. | |
| Ferromanganese in package | | = 5.0 oz. | |
| | | | |
| | | | |

1455.0 oz.

Base..... = r440.0 oz.

CARBON

| Charcoal | 4.5 oz. at. $70 = 3.15 oz.$ |
|----------------|---|
| Nickel | 276 oz. at .005 = 1. 38 oz. |
| Ferromanganese | $6.5 \text{ oz. at} \cdot 06 = .39 \text{ oz.}$ |
| Iron | 1168 oz. at .001 = 1.168 oz. |

6.088 oz. = .42 per cent.

NICKEL

Nickel 276 oz. at .98 = 270.48 oz. = 18.78 per cent.

MANGANESE

Ferromanganese 6.5 oz. at .75 = 4.875 oz. Iron..... 1168 oz. at .oo1 = 1.168 oz.

6.043 oz. = .42 per cent.

ANALYSIS OF STEEL

C P Mn S Si Ni .01 .018 .15 .30 .044 10.17

(4) HARD CHROME STEEL-GRAPHITE POT-COAL HOLES

Melting Time 4 hr.

CHARGE

| Lime, | $\frac{1}{2}$ oz. | |
|--------------------------------|-------------------------|-----------|
| Black oxide of manganese | 2 oz. | |
| 80 per cent. ferromanganese | $1 \frac{1}{2}$ oz. = | 1.5 OZ. |
| 70 per cent. ferrochrome 2 lb. | 2 oz. = | 34.0 oz. |
| Charcoal | $3^{\frac{1}{2}}$ oz. = | 3.5 Oz. |
| Ferrosilicon | $3^{\frac{1}{2}}$ oz. = | 3.5 oz. |
| Iron 88 lb. | = 1 | 408.o oz. |
| Ferromanganese in package | | 7.5 oz. |
| | | |

1458.0 oz. =1440 OZ.

Base.....

CARBON

Ferromanganese.... 9 oz. at .06 = .54 oz. Ferrochrome...... 34 oz. at .07 = 2.38 oz. Charcoal........... 3.5 oz. at .70 = 2.45 oz. Iron..... 1408 oz. at . oo1 = 1.408 oz.

6.778 oz. = .47 per cent.

CHROME

Ferrochrome........ 34 oz. at .65 = 22.1 oz. = 1.53 per cent.

MANGANESE

| Ferromanganese | 9 oz. at .75 1408 oz. at .001 | = 6.75 oz. = 1.408 oz. |
|----------------|----------------------------------|---------------------------|
| | | 8.158 oz. = .57 per cent. |

SILICON

| Ferrosilicon | 3.5 oz. at .50 | = 1.75 oz. | |
|--------------|------------------|-----------------|-----------|
| Iron | 1408 oz. at .001 | = 1.408 oz. | |
| | | | |
| | | 3.158 oz. = .22 | per cent. |

ANALYSIS OF STEEL

| C | $\mathbf{M}\mathbf{n}$ | \mathbf{P} | S | Si | Cr |
|------|------------------------|--------------|-----|------|------|
| . 70 | . 30 | .OI | .02 | .175 | 1.47 |

(5) HARD NICKEL STEEL—GRAPHITE POTS—COAL HOLES Melting Time 4 hr.

CHARGE

| Lime | 2 | oz. | |
|-----------------------------|------|-------|----------|
| Black oxide of manganese | 2 | oz. | |
| 80 per cent. ferromanganese | 3 | oz. = | 3.0 oz. |
| Charcoal | 11.5 | oz. = | II.5 OZ. |
| Nickel 2 lb. | 15.5 | oz. = | 47.5 oz. |
| Iron87 lb. | | = 1 | 392.00z. |
| | | - | |

1454.0 oz. = 1440. oz.

CARBON

Base.....

| Ferromanganese | 3 oz. at .06 | = .18 oz. | |
|----------------|-------------------|-------------|---------------|
| Charcoal | 11.5 oz. at .70 | =8.05 oz. | |
| Nickel | 47.5 oz. at . 005 | =0.238 oz. | |
| Iron | 1392 oz. at . 001 | = 1.392 oz. | |
| | | | |
| | | 9.86 oz.= | .69 per cent. |

MANGANESE

| Ferromanganese | 3 oz. at .75 | = 2.25 oz. |
|----------------|------------------|------------|
| Iron | 1392 oz. at .001 | =1.392 oz. |

3.642 oz. = .25 per cent.

NICKEL

Nickel...... 47.5 oz. at .98 = 46.55 oz. = 3.23 per cent.

ANALYSIS OF STEEL

Mn Ni .87 . 24 3.42

(6) SOFT NICKEL STEEL—CLAY-LINED POTS—COAL HOLES Melting Time 5 hr. 20 m.

CHARGE

| Lime | 1 | oz. | | |
|-----------------------------|---------|-------|-------|-----|
| Black oxide of manganese | 2 | OZ. | | |
| So per cent. ferromanganese | T T | oz. = | 1.0 | ~- |
| Charcoal | - | | | |
| | 5 · 5 | oz. = | 5 · 5 | oz. |
| Nickel 2 lk | b. 10.5 | oz. = | 42.5 | oz. |
| Iron72 ll | b. | = 1 | 152.0 | oz. |
| Ferromanganese in package | | | 4.5 | ΩZ |
| Ferrosilicon in package | | | | |
| I | | | 9.00 | JZ. |
| | | - | | |
| T) | | 1 | 214.5 | oz. |
| Base | | = | I 200 | oz. |

CARBON

| Ferromanganese | 5.5 oz. at .06 | =0.33 oz. |
|----------------|------------------|-------------|
| Charcoal | 5.5 oz. at .70 | =3.85 oz. |
| Nickel | 42.5 oz. at .005 | =0.213 oz. |
| Iron | 1152 oz. at .001 | = 1.152 oz. |
| | | |

5.545 oz. = .46 per cent.

MANGANESE

Ferromanganese 5.5 oz. at .75 = 4.125 oz. Iron..... 1152 oz. at .oo1 = 1.152 oz.

5.277 oz. = .44 per cent.

SILICON

Ferrosilicon...... 9 oz. at .50 =4.50 oz. Iron..... 1152 oz. at .001 = 1.152 oz.

5.652 oz. = .47 per cent.

NICKEL

Nickel...... 42.5 oz. at .98 = 41.65 = 3.47 per cent.

ANALYSIS OF STEEL

| C | $\mathbf{M}\mathbf{n}$ | Sí | P | S | Ni |
|-----|------------------------|-------|------|------|------|
| .19 | .31 | . 367 | .015 | .023 | 3.84 |

An similar analysis would be suitable for castings.

(7) STEEL CASTINGS—GRAPHITE POTS—KRUPP FURNACE

CHARGE

| Black oxide of manganese 3 oz. |
|---|
| Boiler punchings 75 lb. = 1200.0 oz. |
| Heads and gates 25 lb. = 400.0 oz. |
| Washed metal 10 lb, = 160.0 oz. |
| 80 per cent. ferromanganese 1.5 oz. |
| 60 per cent. ferrosilicon |
| |
| 1771.5 oz. |
| Base = 1760. oz. |
| CARRON |
| CARBON |
| Punchings 1200 oz. at .002 = 2.40 oz. |
| Gates |
| Washed metal 160 oz. at .03 = 4.80 oz. |
| Ferromanganese 1.5 oz. at .o6 = .o9 oz. |
| |
| 10.29 oz. = $.58$ per cent. |
| SILICON |
| Ferrosilicon 10 oz. at $.55 = 5.5$ oz. |
| Gates |
| Punchings 1200 oz. at .001 = 1.2 oz. |
| 2 |

7.5 oz. = .43 per cent.

| Ferromanganese | 1.5 oz. at .75 | =1.125 oz. |
|----------------|------------------|---------------|
| Punchings | 1200 oz. at .002 | = 2.400 oz. |
| Gates | 400 oz. at .003 | = I . 200 OZ. |

4.725 oz. = .27 per cent.

MELTING TIMES AND ANALYSES OF HEATS

MANGANESE

| Time | C | Si | Mn |
|------------|------|------|------|
| 5 hr. 45 m | .71 | . 26 | .34 |
| 5 hr. 45 m | .61 | . 25 | . 27 |
| 5 hr. 30 m | . 75 | .28 | • |
| 5 hr. 55 m | .62 | . 20 | . 29 |
| 4 hr. 45 m | .67 | - | . 29 |
| 6 hr. 15 m | .68 | .30 | . 28 |
| 5 hr. 35 m | .80 | .34 | . 30 |
| 5 hr. 35 m | .80 | .33 | . 38 |
| J JJ | . 80 | . 27 | . 36 |
| Average | | | |
| Average | .71 | . 29 | . 36 |

(8) SAME AS (7), EXCEPT THAT FERROMANGANESE = 3 OZ. FIGURED ANALYSIS

| C | Si | Mn |
|------|-----|-----|
| . 59 | .43 | .33 |

MELTING TIMES AND ANALYSIS OF HEATS

| Time | C | Si | Mn |
|------------|------|------|------|
| | . 84 | . 28 | .31 |
| 6 hr. 40 m | . 66 | . 29 | . 32 |
| 5 hr. 40 m | .77 | . 31 | .35 |
| 4 hr. 50 m | 71 | .31 | .34 |
| | | | |
| Average | .75 | .30 | -33 |

(9) THE ANALYSES OF 25 HEATS OF CHARGES (7) AND (8) GAVE RESULTS AS FOLLOWS:

| | P | S |
|---------|------|------|
| Lowest | .019 | .016 |
| Highest | .062 | .088 |
| Average | .033 | .063 |

Three points of crucible steel foundry practice are illustrated in these figures—the use of washed metal and open-hearth scrap as raw materials, the high carbon often carried to make the charge melt easily, and the high sulphur and phosphorus content of some of the steel made.

Not many years ago, a steel foundry was started in this country using a so-called "new process" invented abroad, by which steel was made containing less than .20 per cent. carbon. This process consisted in nothing more nor less than the use of a Siemens gas furnace and clay pots, in which mild steel scrap was melted with the addition of enough silicon and manganese to make the metal set sound in castings. The cost of the process, of course, was extreme, more especially as the furnaces could not stand up long to the extreme temperatures necessary to make steel of this analysis properly fluid. The matter is mentioned here to illustrate the comparative unfamiliarity of many people with the possibilities (and the reverse) of the crucible process.

ALLOY STEELS

The crucible process is well suited to the manufacture of ordinary carbon steels, except those very low in carbon; and of nickel, chrome, nickel-chrome, tungsten and molybdenum steels. Examples of most of these are shown in the tables above.

Manganese steel can be produced in crucibles, by charging a mild steel mixture that will melt at about .20 to .25 per cent. carbon in clay-lined pots, and mixing this steel with proper amounts of ferromanganese, melted separately in graphite pots. It is not possible to melt the whole charge together, because, as already noted, mixtures high in manganese cut deeply into crucibles and absorb a great deal of carbon. When an attempt is made to melt wrought iron, or soft steel scrap, and ferromanganese together in a graphite pot, the absorption of carbon is so great as to result in a steel containing 2 or 3 per cent. carbon, quite useless for the purposes for which manganese steel is made. Even in clay-lined pots, the cutting action is so severe as to eat through the clay lining and cut deeply into the graphite, so that the carbon of the steel will be far too high.

For the same reason manganese steel sink heads, gates and other scrap cannot be re-melted in graphite or clay-lined crucibles, so that the scrap has to be sold or used up in open-hearth furnaces. Clay crucibles could be used for making the steel and re-melting the scrap, but clay crucibles as already noted are awkward to handle, and the cutting action of the scrap is so severe as to wear out pots very rapidly.

The cost of manganese steel made by the crucible process is so high that it is not practicable to produce the steel in this way, except for certain special products of small size—and even for these castings other processes are more suitable.

CHAPTER IV

THE BESSEMER PROCESS

The Bessemer process for converting pig iron or its equivalent into steel was invented by Sir Henry Bessemer in the 1850's, and was the first method ever developed for the production of steel in large heats and at a low price. It is an indirect method—that is, it converts pig iron, the first product made from the ore, into steel—and owes its cheapness to the tremendous tonnage that can be turned out with a comparatively inexpensive installation.

The process consists in blowing air through a molten mass of pig iron, high in carbon, silicon and manganese, contained in a cylindrical vessel lined with refractory material and open at the upper end for the escape of the gases. The oxygen of the air oxidizes the silicon, manganese and carbon very rapidly, the silicon and manganese forming SiO₂ and MnO, which unite with FeO produced by the oxidation of some of the iron, and with additional SiO₂ from the lining of the vessel to form a slag; the carbon escaping into the air as CO and burning at the vessel's mouth to CO₂, producing an intensely bright flame. In from 8 to 15 minutes the impurities in the steel are all oxidized, and iron begins to burn rapidly, producing a great deal of FeO, some of which is absorbed by the metal, the rest going into the slag.

The combustion of the silicon to SiO_2 , and to a less degree that of manganese to MnO, produces a great amount of heat. That of carbon to CO adds only slightly to the heat of the steel. The heat produced by the combustion of these elements raises the temperature of the molten iron the amount necessary to maintain it fluid when the impurities are all removed and a bath of metallic iron remains.

It was the FeO absorbed by the steel at the end of the blow that hampered the early experiments of Bessemer, and for a time rendered the success of the process doubtful. The metal when tapped from the stationary vessels first used, was so heavily oxidized and full of gas that it was "wild" in the moulds, full of blowholes, weak, and generally worthless. Nevertheless, plants were built to produce

Bessemer steel and constant efforts made to overcome this difficulty. In Sweden, the process was a success practically from the start. and many plants were erected that produced excellent steel without difficulty. Upon investigation, it was found that the Swedish pig iron used for the process was so high in manganese that much of this element remained unburned till near the end of the process; and that as the Swedes had adopted the procedure of stopping the blow when about the amount of carbon was left in the metal that was desired in the finished product, much manganese remained in the blown steel. It was thought, rightly as it proved, that this manganese counteracted the injurious effects of the oxidation of iron. and that in blowing pig iron low in manganese, the addition of spiegel (pig iron containing 10 to 20 per cent. of manganese) might overcome the difficulties that were being experienced in English practice. The surmise proved correct, the low manganese pig iron blown as usual and then treated with molten spiegel to add manganese (and at the same time the desired carbon), was found to produce excellent steel, and from that time on the success of the process was assured. In 1856 Sir Henry Bessemer read his paper on "making wrought iron and steel without fuel," and acknowledged the necessity of manganese—Mushet obtained a patent on the use of manganese the same year.

The development of the process for making steel of all grades from soft structural steel to medium hard steel for rails, and even high carbon steel for tools, was very rapid. Inventors in all countries vied with one another in improving the mechanical devices used in the process, and increasing the size of the vessels and the output. The vessel was mounted on trunnions, whereby the speed of handling was greatly increased; in place of melting the pig iron in cupolas, the fluid pig from the blast furnaces was kept hot for use in the Bessemer plant in a "mixer" (a tilting furnace of large capacity); the spiegel necessary for deoxidizing and recarburizing was melted in cupolas instead of reverberatory furnaces; and the cranes, casting pits and other apparatus for getting the steel into the ladles and thence to the moulds were greatly improved. At the same time the size of the vessels was increased, until in place of the 2- and 3-ton vessels first used, the standard steel works size in America to-day is from 10 to 20 tons.

In the manufacture of castings, the large bottom-blown vessel has been used to a certain extent from the first, but the application of the Bessemer process to the steel foundry soon involved the design of small units, and was hampered by the difficulty of producing steel in small heats that was hot and fluid enough to be run into light castings. In the efforts that were made to overcome this difficulty two lines were followed. The first was to retain the bottom-blown type of vessel and find means to produce higher temperatures by changes in the volume of air blown through the metal, that is, speeding up the process; by changes in the amount of heat-producing elements contained in the pig iron; and by adding heat-producing elements at some stage or stages of the blow. The second was to utilize the principle soon invented for warming up cold heats in the ordinary vessel, of blowing at or over the surface of the bath by tipping the vessel forward, and attaining extra temperature by the combustion of iron and the burning of CO to CO₂ within the vessel. This led to the perfection of the side-blown vessel.

It was soon found that of the manganese added to Bessemer steel, a part was found chemically combined in the steel, a part disappeared. Moreover, within limits, the more manganese was added, the greater was the actual weight of manganese lost. As we now know, the manganese reduces the FeO in the steel by the reaction

$$FeO + Mn = MnO + Fe$$

and the manganese oxide goes largely into the slag. Part of it is retained in the steel, but its influence is not so harmful as that of FeO.

A second function of the manganese is to unite with the sulphur of the steel, which normally exists as sulphide of iron, FeS, to produce sulphide of manganese, MnS. Owing to its existing in the steel as more or less rounded globules, instead of in strings between the crystals as FeS does, MnS does not make steel "red short" (i.e., weak and brittle when hot) to anything like the same degree as does FeS; and as iron does not dissolve the sulphides of other metals readily, some of the MnS floats out of the steel. These two functions of manganese are the basis of modern steel making.

A point to be clearly remembered in this connection is that, as already stated, more actual weight of manganese added increases the weight of manganese lost, that is the manganese converted to MnO and slagged. This means that the reaction between manganese and oxide of iron does not readily complete itself, owing to the tenacity with which metallic iron retains small quantities of its own oxide in solution; and that in order to force the maximum

amount of FeO to react with the manganese, we must have a considerable excess of the latter present. It is almost as if the manganese were a weight, the iron a sponge, and the FeO the water held in the interstices of the sponge. By getting a 4-oz. weight on the sponge we force out part of the water, putting on 4 oz. more expels still more water, and so on. We never can force all the water from the sponge by pressure; and by the reducing action of manganese alone, especially in the Bessemer process in which the manganese has but a short time to act, we can never expel all the FeO from the steel. A proportion remains, which depends largely upon the amount of manganese added, and the time allowed for it to act. It is to this residue of dissolved FeO, as well as to the gases absorbed and never completely expelled, that the inferiority of Bessemer steel is attributed.

In proof of the foregoing statement, it is only necessary to instance the variation in the losses of manganese in the manufacture of carbon steel and of manganese steel. In the former, if we add say .75 per cent. manganese, we will get usually between .60 and .70 per cent.; if we desire 1.25 per cent., we often have to add about 1.45 per cent. In making manganese steel, it is often true that of 14 per cent. added to the steel, only about 12 to 13 per cent. remains.

The deoxidation of the steel, moreover, can be more completely secured by the addition of ferrosilicon, and a "final deoxidizer" such as silicon-calcium, ferrotitanium, aluminum and the like. It is as well to remind the reader, however, that during the last 10 years many of these "final deoxidizers" have been put on the market that proved to be of very little use, and which are no longer obtainable. One is tempted at times to entertain the uncharitable suspicion that when the promoters of a new alloy can find no beneficial effects of the metal on the physical properties of the steel to which is added, they proceed to put it on the market as a "final deoxidizer."

The advantages of the process of the production of light castings are:

- 1. Cheapness of installation.
- 2. Cheapness of steel produced.
- 3. Large output per dollar invested.
- 4. High temperature of steel.
- 5. Suitability for intermittent operation.
- 6. Small heats at short intervals.

The installation costs of a small Bessemer plant are low, especially if a side-blown vessel is used, as is almost always the case, as

the chief items of expense for the side-blown plant are vessel, cupolas, positive pressure blowers and motors to drive them. No very expensive foundation work is required, and the equipment is quite simple. The bottom-blown shop requires, instead of positive pressure blowers, a blowing engine of considerable size and boilers to supply it with steam, making the equipment more costly than that of the side-blown shop. In the introductory chapters, figures have been given that show the comparatively low cost of the steel per ton, and the great output per dollar invested. The loss of metal in blowing is higher in the side-blown converter than in the bottom-blown vessel; the loss in melting and blowing is about 16 to 18 per cent. in a shop using side-blown converters, and about 12 to 14 per cent. with bottom-blown vessels. In some cases this considerably affects costs. The side-blown vessel produces metal that is considerably hotter than that of the bottom-blown, but either if properly handled can be counted on to turn out steel that will pour into light castings without undue difficulty.

Intermittent operation of a small Bessemer plant is the rule rather than the exception. If the cupolas are run every day, and in the case of the bottom-blown vessel, if the vessel is run every day, the operation of the shop will be more uniform. The fuel required to heat up the vessel and cupolas does not cost much, however, and as the installation costs, especially of small side-blown shops, are quite low, many such shops are regularly run only part time when business is dull.

For a foundry making small castings, the light heats of the Bessemer vessel, coming at frequent intervals, are an advantage in pouring for two reasons. First, the metal can be brought very hot to the moulds and all poured before it cools off. Second, the work of the moulders is not greatly hindered by the pouring, since only part of the shop is invaded by the pouring gang at each heat and the men elsewhere can go on moulding undisturbed.

The disadvantages of the small vessel in steel foundries are: (1) Quality—the steel not being as good as crucible or electric steel, with which it competes, nor on the average as good as acid openhearth steel. (2) Small heats—the pouring of castings of over 3 or at most 4 times the gross weight of the steel that can be produced at one heat is extremely difficult with a bottom-blown vessel, and over twice or three times the weight of a heat with a side-blown vessel. This disadvantage, however, is more apparent than real, since it affects only the production of heavy work and the small converter is

seldom able to compete successfully with the open-hearth foundries in heavy carbon steel castings. In some kinds of specialty work, where the specialty is an alloy steel made in castings of all weights from a few ounces to 10 to 12 tons, this disadvantage becomes a real one, and makes it probable that the ideal shop of this kind would have both open-hearth and Bessemer equipment, using the latter for the special steel in small sizes, and the former for the heavy special steel orders and for a miscellaneous trade in machinery castings of the better class. (3) Flexibility, by which is meant the ability to produce readily several different kinds of steel in small lots. In Bessemer practice, the only way to make several kinds of steel from one heat is to put molten recarburizers in two or more ladles and add a weighed amount of the soft metal from the vessel. When a very sticky slag is made (in the Bessemer process the slag is sometimes semi-solid), it is possible to hold back the slag with a skimmer, and thus get the correct amount of metal in the ladle. When a fluid slag is made, it becomes well-nigh impossible to weigh the metal accurately.

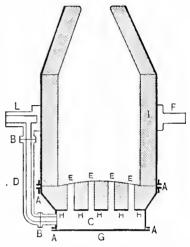
This practice, moreover, cannot be carried very far in Bessemer work because the chilling of the steel in the vessel is so rapid that not more than two compositions of steel are readily made, and if the heat has been blown too cold, not even two.

The Bottom-blown Vessel.—The bottom-blown converter of small size is used in only one shop in this country, so that there is but one general type made. This is a plain cylindrical vessel of about 3 tons capacity, with tapered nose, built all in one piece except the bottom and wind box, which are arranged to be readily removed from the body of the vessel. The vessel is mounted on trunnions, is revolved by a hydraulic cylinder with a rack working on a pinion on one vessel trunnion, or by an electric motor. The blast main enters through one trunnion, goes down the side of the vessel, and is attached to the wind box through a coupling readily broken when the bottom is removed. The bottom and wind box are permanently fastened together, and clamped to the body of the vessel in such a way as to be readily taken off when a new bottom is needed.

The vessel proper is generally lined up with flat slabs of so-called mica schist (a very siliceous gneiss or schist) or of silica rock, beginning at the nose and working upward (the vessel being swung into position with nose down), until the whole is filled. The lining in the main part of the vessel is made about 10 in. to a foot thick, the slabs are wedged tightly in place, breaking joints carefully, and the whole

well plastered with siliceous ganister (old silica brick or silica sand and pebbles ground up with enough clay to make the mass plastic). The bottom of the converter is provided with holes between wind box and lined space, on which the clay tuvères are set. These are bricks

some 2 or 3 ft. long, each pierced with a number of small holes about ½ in. in diameter through which the air passes. The outside of the bottom is lined with mica schist or rock of the same sort as that used in the body of the vessel, the space around the tuyères is partly filled up with large bricks set on end, and the remaining space is then packed with ganister ground very dry, shoveled in a few inches at a time. and tamped down with hot moulders' rammers. The whole is brought to a slightly convex for bottoms should be selected tuyères; I, lining. with care, as the life of bottoms



surface, and is then dried out in an ordinary flask oven. The lugs; B, joints in blast pipe; L, F, truntuyères, bricks and ganister used nions; D, blast pipe; G, wind box cover; C, wind box; H, bottom plate; E,

is short enough at the best, and to use tuyères that melt or slag away rapidly greatly shortens this life.

The number of tuyères used for a 3-ton vessel is such that the total tuvère area is about 6 to 10 sq. in. If each tuvère has seven holes $\frac{3}{8}$ in. in diameter, and nine tuyères are used, this gives 6.95 sq. in. tuyère area. By varying the size of holes in one or more of the tuyères, and sometimes by omitting one tuyère, this area can be varied to meet the requirements. In general, with a given equipment, to reduce the tuyère area means slower blowing and less "slopping." The pressure of the blast can be changed to accomplish the same ends. The blast is supplied at a pressure of from 15 to 30 lb. per square inch.

The life of a vessel lining should be from 6 months to 1 year. The composition of the metal charged in the vessel, by its influence on the slag produced, affects the life of linings, and especially of bottoms, very considerably. A high silicon, low manganese mixture produces a very siliceous, sticky slag, which cuts the linings hardly at all, but wears away tuyères and bottom rapidly; while higher manganese produces a slag rich in oxide of manganese, fluid, and greedy for silica, which cuts the linings severely. Such a mixture, however, does not use up the bottoms as fast as the high silicon

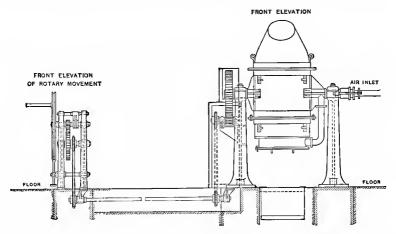


Fig. 6.—Walrand-Lègenisel converter. Front elevation. From Journal of the Iron and Steel Institute.

metal. In general, it is better to proportion the ganister used to give the greatest possible life of linings and bottoms with a given iron mixture, rather than to suit the mixture to the lining. This is especially the case where scrap can be used as a substitute for part of the pig, resulting in a considerable saving, though at the expense of linings on account of manganese in the scrap. It costs so little to line a vessel that the decreased life of lining is much more than offset by the saving in materials.

Bottoms will last from 15 to 35 heats—the latter figure is a high one. Frequently, on account of the delay incident to changing bottoms in the middle of a day's run, and the cold steel inevitably blown in the first one or two heats on a new lining, a bottom which would make several more heats is removed at the end of a day's run, rather than leave it on for the first few heats the next day. Of course, if there are two vessels, bottoms should be used until thoroughly worn out.

Between heats the nose of the vessel is scraped out with an iron bar to remove loose slag, and places in the sides that are cutting out are plastered with ganister. Before starting up in the morning the cut places are similarly patched, if the lining is being cut out by the slags. With very siliceous slags the tendency is sometimes to build up the lining, and slag occasionally has to be cut away from the sides of the vessel with bars and sledges.

When a bottom is nearly used up and one or more tuyères are getting short, it is necessary to blank off a few holes in the tuyères, or sometimes a whole tuyère, by ramming the holes full of ganister, to prèvent the iron running through the holes into the wind box. For this purpose the cover plate of the wind box is made readily removable, so that between heats the plate can be taken off and tuyères blanked off from the bottom. Holes in long tuyères that get stopped up are at the same time poked out with small bars.

Bottoms are put on by turning the vessel upside down, shoveling a good layer of ganister on the lining where the joint comes, setting on the bottom and tightening up the bolts, at the same time ramming the ganister tight with hot moulders' rammers. Coke dust is used to keep the ganister from sticking too tightly to the lining, lest it pull the latter off in charging bottoms.

Before blowing steel, the vessel has to be brought up to a good yellow heat. This is generally done by building a wood fire in it, turning on a gentle blast, and then dumping in lumps of soft coal. The fire is kept going 3 or 4 hours, and serves very well.

It is hardly possible to give the crew needed to handle the shop, as this varies considerably with the tonnage made, furnaces used for melting recarburizers, etc. Omitting the firemen on the boilers, and the engineer of the blowing engine, the men needed are:

Blower—generally foreman of the whole shop. He works, when blowing, in the "pulpit" where he can see the flame of the vessel and as much as possible of the shop. In the "pulpit" are the levers controlling the blast and the revolving mechanism of the vessel, engineroom push buttons or whistle cord, etc.

Vessel foreman-in charge of vessel and ladles.

Ladle man and two or three helpers, who also assist on the vessel.

One (or two) men on vessel charging platform, who may be part of ladle gang.

Cupola tender and helper.

One cupola charger—if mechanical charging be used.

Yard gang, who handle raw materials for cupolas, and charge the latter in many shops.

Physic furnace foreman and helpers, (when molten recarburizers are required).

For vessel, ladles, etc., and cupolas, 14 or 15 men should be able to take care of 15 or more heats a day, charging the cupolas by hand.

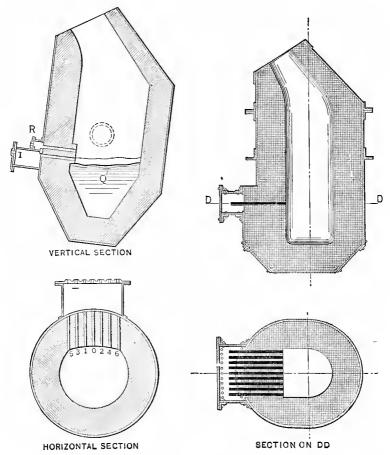


Fig. 7.—Tropenas converter. From Transactions of the American Institute of Mining Engineers.

Fig. 8.—Stoughton long-tuyère converter. Vertical and horizontal section. From Transactions of the American Institute of Mining Engineers.

The Side-blown Vessel.—The side-blown vessel, which is the type almost universally used in steel foundries in America, differs from the bottom-blown chiefly in the construction of the vessel, and the

pressure and method of application of the blast. It can be successfully used in smaller sizes, 2 tons being the most common capacity. In the original type of side-blown converter, the tuyères were placed around the sides of the vessel, somewhat below the surface of the metal. In consequence, a fairly high-blast pressure was required. These vessels may be considered as merely a modification of the bottom-blown type. The heavy wear on the lining at the tuyères, coming on the sides instead of on the removable bottom, is a great disadvantage because it renders necessary repairs to the lining itself every 20 to 30 heats. These converters are now little if at all used in America and will not be considered in this work.

The true side-blown vessel is one in which the blast is introduced through tuyères in one side of the vessel, above or at the surface of the bath. The Tropenas vessel was one of the first of this design to be widely used in America. In its original form it was provided with two rows of tuvères, the upper being designed to introduce extra air and burn CO to CO2 more advantageously than by blowing an excess of air through the lower set. The extra set, however, proved to be difficult to manipulate and are now seldom provided. For some years, the bath in this vessel was made very deep, with the object of preventing undue oxidation of iron. To-day, however, practically all makers of side-blown vessels have adopted the shallow bath, and there is little difference in the design of the converters sold by various firms. In placing a tamped lining in the Tropenas vessel, a collapsible form is used. The tuyères, which are commonly of brick, are comparatively short, and require replacing every 20 heats or so.

In recent years, several designs of two-piece vessels have been placed upon the market, in which either the top or the bottom half is made removable and is clamped on much as are the bottoms of bottom-blown vessels. The original converter of this type was designed by Bradley Stoughton, and is arranged so that the bottom half, with the tuyères and wind box can be removed. There are other designs on the market to-day, in some of which the top half is removable. The removable feature facilitates relining, as a solid wooden form is used for each part in placing tamped linings, and is said to allow of the use of a single vessel for continuous operation; indeed shops have been operated with but one converter. In other foundries though the removable feature is considered an advantage, yet two or more converters are used. The difficulty of matching a partly worn top with a newly lined bottom is given as the reason for

using several vessels, but the fact that foundries have successfully operated with but one converter proves that it can be done. Further features of the Stoughton converter for which advantages are claimed are the use of iron or steel pipes for tuyères, held in place by a casting on the side of the vessel, in which they have a long bearing, some 9 in.; the thickening of the lining at this point, whereby longer tuyères are used; and increased height of vessel, whereby slopping is minimized. The placing of the tuyères is much simplified by the use of the solid bearing for them, and as they burn off they can be pushed forward a little and the lining patched around them. By the use of a thicker lining at the tuyères it is claimed that 25 to 30 heats can be made without putting in new tuyères. The removable bottom allows of extensive repairs to the lower half of the lining without disturbing the upper half.

The linings used in these converters are of three types: silica rock, (ganister rock), or mica schist slabs set in the converter in the same manner as the lining of a bottom-blown vessel; silica brick linings made up of special shapes; or a mixture of silica sand (and pebbles) and clay tamped in place around a wooden core to give the shape of the inside. When brick linings are used, the tuyères are generally in the form of special bricks. In stone or tamped linings the tuyères are made by moulding the lining round wooden forms (which are afterward pulled out), by the use of brick tuyères, or by using iron or steel pipes for the purpose which are left in place. In making a tamped lining one course of brick is generally used next to the shell, and the bottom is built up of brick for one or two courses.

The tuyères are generally arranged in fan shape, in order to cover the whole cross-section of the converter with the blast. Minor repairs in the way of patching the lining, especially at the tuyères, have to be made for every run. In most shops the vessel is used only every other day, making from 7 to 15 heats, and the patching done on the idle days. The lining at the tuyères is generally patched from the outside by cutting it away at this point and replacing tuyères and lining through the wind box opening. For this reason it is commonly necessary in order to blow steel every day, to have two or three vessels, since the converter cools off too slowly to enable the men to work at it and have it ready for use every day.

To overcome this difficulty, the Tropenas Company has introduced, in small sizes, a drop-bottom converter. In this design, the bottom is closed by a swinging door, such as those used on cupolas, which is dropped after the day's run, and the bottom punched out. With

both ends of the vessel open it cools quite rapidly, and repairs can be made to the lining each morning without difficulty.

The top section of the lining of two-piece vessels lasts several months with only a little patching. In one shop, running a vessel every other day, making about 8 heats a day or 24 a week, a ganister lining is said to last about 9 months.

The side-blown vessel is heated up every day before steel is blown, sometimes with a coal fire, but more often by means of an oil burner introduced through one of the tuyères.

The blast used in these converters is at a pressure of 3 to 4 lb. per square inch, occasionally 5 lb. In order to furnish blast at this pressure in the necessary quantity, a positive pressure blower is required.

Special Types of Converters.—One special design of small converter deserves notice here, before the general description of Bessemer practice is taken up. This is the Stock converter, in which the pig iron and scrap are charged cold and melted with oil burners introduced through the tuyères, after which the vessel is turned up and the metal blown. During the melting period the hot gases from the oil burners are taken off through an "economizer" or brick chamber containing iron pipes, which is opposite the nose of the converter while oil is being burned. When the metal is melted, the vessel is turned up to blowing position and the blast is taken through the pipes of the economizer, whereby its temperature is considerably raised. By the use of hot blast a very high blowing temperature is said to be attained. The advantages of this type of vessel are the high temperature of the metal, and the fact that in oil melting there is little absorption of impurities from the fuel, if low sulphur oil is used, whereby the increase of sulphur in the iron, which is often so troublesome in regular practice, is avoided. This vessel has a distinct field of its own—the production of a very small tonnage of small steel castings as an accessory to shops not contemplating the production of steel in quantity, in localities where coke is costly. For this field its cheapness of installation commends its To attempt to compete with users of regular types of converters in localities where coke is cheap is quite another matter. To begin with, the melting of the metal will be considerably more costly by this process than by cupola melting, when enough steel is to be made to give the cupola a good run. In the second place, the Stock vessel will turn out steel at the rate of only about one heat every 2 hours. Compared with the regular vessel which produces two or three heats

an hour, therefore, the Stock converter is at a disadvantage, as it would require from four to six of these vessels to equal the production of one converter of equal size supplied by a cupola. As the economy in installation is confined to cost of cupola and its blowers, we can readily see that for considerable tonnages this saving will vanish, since one cupola with blower or fan can be installed more cheaply than three to five extra vessels. These disadvantages have discouraged the adoption of the process in this country.

The field for the Stock converter will therefore be found in places where coke is costly and oil cheap. Under such conditions oil melting may be cheaper than cupola melting, but even then the disadvantage of being obliged to use a number of vessels to attain high output will remain. To take advantage of cheap oil and costly coke under these circumstances it might be more economical as far as melting costs go, for the production of a considerable tonnage of small castings, to install oil-burning melting furnaces and use them to supply a single converter of the usual type. The first cost of such an installation, however, would be greater than that of a battery of Stock converters, and if a regenerative furnace were used it could not be run intermittently, as converter plants can.

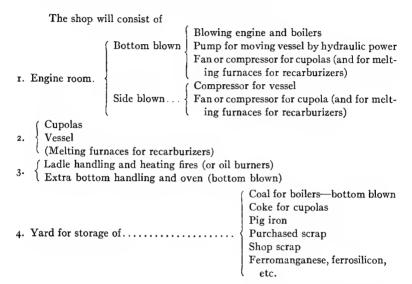
Bessemer Electric.—Some years ago, a combination of Bessemer converter and electric furnace was proposed, in which cheaply melted cupola metal was to be blown, and than refined by electric current. The electrodes were to be introduced through the side of the vessel after it was "turned down." The idea, of course, was to save the loss of heat in transferring metal from converter to electric furnace, where "duplexing" was to be the regular practice.

It seems hardly necessary to say that such a piece of apparatus will be neither a good converter, nor a good electric furnace, and that no progress has been made in developing it.

LAY-OUT OF BESSEMER SHOP

The lay-out of a shop depends so much upon the conditions in each case that it is not easy to give more than the general rules that apply to the subject, and many of them are rather negative than positive, stating rather what not to do than what to do. For convenience we will assume that our shop is of fair size containing say a 3-ton bottom-blown vessel or two 2-ton side-blown vessels, and that we shall produce about 12 to 20 heats per day; for larger outputs, two bottom-blown or three to four side-blown vessels are

desirable. This comparatively large production is taken as a basis, since a shop making but a few heats a day, or a few heats a week, does not involve the problems in economical handling of material that demand attention in laying out the plant.



In the general arrangement of the shop, care should be taken in the first place to see that the units are so placed that they will not be in each other's way when the increasing ouptut of the shop calls for extensions. In a bottom-blown shop, with its bulky installation of boilers, blowing engine, etc., this heavy machinery cannot readily be moved to allow for errors in original placing, and care should be exercised to locate it so that the other units of the plant can readily be extended without being interfered with by a badly placed engine room. At first sight this seems a simple matter, but when it is considered that the blowing engine must be fairly close to the vessel (because the engineer must see what is going on), the problem becomes more difficult. As the large blowing engine of the bottom-blown vessel is generally steam driven it will frequently be economical to install steam-driven pumps and to provide hydraulic power to rotate the vessel, and other small steam equipment.

The engine-room equipment for a side-blown vessel is so much less bulky that it is far easier to place to advantage The compressors are generally driven by electric motors, and occupy relatively

little space. The vessel is most often turned over by an electric motor, and this still further reduces the total of equipment.

The cupola or cupolas for melting the vessel charge should be so placed as to most easily transfer the metal from them to the vessel. In most shops the cupolas are tapped near the ground and the metal brought to the vessel by lifting the ladles with the overhead crane that serves the converters and pouring floor. In some of the early converter shops the cupolas were elevated, and their spouts discharged directly into the vessels. This led to trouble in blowing, due to the cupola slag getting into the converters with the metal.

When many heats are to be blown per day, it would seem desirable to place the cupolas at a higher level than the converters, and tap into a ladle on a vessel charging floor. This ladle may be swung with a jib crane, or mounted on trunnions and turned by a wheel, gear and pinion, running the metal through a spout into the vessel. The trunnions might with advantage be mounted on a platform scale to weigh the metal for each heat.

This arrangement prevents delays in crane service and is especially desirable when several consecutive heats are to be blown for one casting, as delays in crane service may then retard the rapid production of the heats so essential to success. It may be objected that the extra height to which the pig iron and coke have to be hoisted is a disadvantage; but they have to be hoisted anyway, and if an elevator is to be used a few feet of extra lift do not materially affect the cost of hoisting. If the material is to be delivered to the cupola charging platform by an overhead crane in the stock yard, the extra height of crane runways needed for the higher cupolas would be a more serious matter, but in the majority of small steel foundries other means of yard handling are generally cheaper than an overhead crane so that this objection will not often be a valid one.

To facilitate the use of cupola metal for recarburizing high carbon heats, if the shop were arranged in this way, a swinging cupola spout would be of great advantage. By this means small ladles on a platform scale could be filled from the cupola with the proper amount of metal, and their contents poured into the vessel; or if the recarburizing were done in the ladles, the latter could be set upon the scale and filled direct from the cupola. Either arrangement would be more convenient than taking metal from the ladle used to weigh vessel charges. The bottom-blown vessel is swung into a horizontal position for charging, and generally is charged when turned over in one direction, and poured when turned the other

way. This would necessitate the use of two spouts if recarburizing were to be done in the vessel, one to run in the blowing charge, and one to run in recarburizer. The side-blown vessel, on the other hand, generally is charged and poured on the same side. Hence if the recarburizers were to be added to the vessel, they could be run in by means of the same spout as that used to pour in the blowing charge.

Melting furnaces may or may not be installed for premelting "physic" (that is recarburizer), depending largly upon the grade of steel to be produced. If much hard steel, or steel containing a high percentage of alloys, is made, they are frequently a necessity, though at times the use of fluid metal from the main cupolas is all that is necessary.

The melting furnaces used for recarburizers should be of a type that can be operated intermittently, because of the generally intermittent, or at least single-turn operation, of the shop as a whole. Depending largely upon the kind and quantity of metal to be melted, we may use:

- 1. Small cupola.
- 2. Air furnace (coal or oil fired).
- 3. Crucible furnace.
- 4. Electric furnace.

The question of choice among these methods of melting is discussed in the section on the use of recarburizers.

If furnaces are used for melting recarburizers they should be placed so that they can readily pour their metal into the steel ladles, set on scales; or into a small ladle that can be poured into the vessel. When cupolas are used, if it is desired to recarburize entirely in the vessel, they can be placed so as to top on the level of a vessel charging floor, as the larger cupolas can be. As a rule, however, they are set at the foundry-floor level, and tap into the steel ladles, or into smaller ladles which are carried to the converter by hand or by crane, and poured into the vessel.

The main working floor should be as roomy as circumstances allow. Nothing is more fatiguing and exasperating to the men than a crowded shop where they have no elbow room and no place to jump to when the vessel "slops." As has already been pointed out in discussing crucible steel foundries, the greatest pains should be taken to provide means of throwing open in summer all places where men have to work—main floor, vessel platform, cupola tapping and charging platforms, physic furnaces, etc. Neglect of this important

feature of design frequently adds considerably to the labor cost of shops when they grow to a good size and handle metal in bulk, since a hot stuffy working place simply means that the amount of work each man is physically able to accomplish per day is reduced and working forces are artificially enlarged.

Working space for lining and drying out ladles, fixing up stopper rods, etc., should be ample. Ladles may be dried and heated up with hard coal or coke fires, or oil or gas burners. The choice will be largely a matter of the relative cost of the fuel. In bottom-blown shops several extra bottoms are kept on hand, and one or two are always ready, lined up and dried out in a flask oven located conveniently to the melting shop. Storage space for tuyères, ladle nozzles and stoppers, sleeve brick for stopper rods, etc., should be conveniently located.

In converter shops, a pit as a rule is provided in front of the vessel. into which the ladle is lowered to be filled. This is generally made with steep sides, lined with concrete, and large enough to hold the slag that spills from the ladles and is scraped from the converter during the day, and still leave room for the ladle. These pits are necessary when the converter is not provided with a charging floor, and the cupolas are set at ground level, as the converter would be at a most inconvenient level if set high enough to allow the ladles to be filled without the use of a pit. Were the cupolas arranged to tap at a higher level and the vessel filled from a charging floor, a deep pit would be unnecessary.

These concrete pits have always seemed to the author a poor feature of the lay-out of the ordinary converter shop. In the first place they are so steep sided that if for any reason a heat has to be poured into the pit, the metal is hard to remove. In the second place, they have to be covered with planks to allow the men to get close to the vessel when pouring in the charge and the recarburizers; and in the third place, men have a way of falling into pits.

Unless it is thought advisable to elevate both cupolas and vessel, however, a pit is a necessity; and designers apparently do not believe in elevating their cupolas.

Storage Yard.—The storage yard and raw material handling generally force themselves unpleasantly upon the attention of the management as soon as the shop begins to turn out a large tonnage. Too often it is then found that the whole shop has been so located that changes in methods of storing and handling material are difficult to make, when a little forethought in the beginning would have

taken care of the whole matter. Means are generally provided to dump coke for the cupolas, and coal for the boilers, etc., from trestles, and to get them to the places where they are wanted without too much expense. It is in the handling of pig iron, scrap and alloys that the trouble generally comes. A small shop as a rule cannot afford to use an overhead crane or a locomotive crane with a magnet for unloading raw material, and depends upon hand labor for this work. This is quite right and proper, but the error is too frequently made of failing to forsee that eventually crane handling must take the place of hand labor and to make provision for the time when this is to take place. However the yard be arranged, everything that can be dumped from cars into bins or piles, such as coal, coke, ferromanganese (in quantity), etc., should eventually be so handled; and pig iron and scrap should be unloaded with a magnet, either with an overhead crane or with a locomotive crane.

Though the materials for the cupola charges, as well as coke, generally have to be loaded on cupola charging cars by hand, nevertheless the number of times they are handled should be the smallest possible. The author has seen cases where some of the pig iron is picked up seven times by hand before it goes down the cupola, owing to inadequate and badly arranged storage space, which necessitates piling the iron, and sometimes moving a pile from place to place. Generally the cars or lots of pig are kept separate, to facilitate control of cupola "mixtures;" but if adequate space is provided, the pig can be left in heaps where it is unloaded, and taken from these heaps to be put on the charging buggies.

In most shops the pig and scrap are sent to the cupola charging floor by elevator in weighed lots on buggies, and charged by hand. The coke and limestone are generally kept in piles on the cupola charging floor and weighed out for charging as needed, more being sent up from time to time. Many large grey-iron foundries use cupola charging machines with success, and the economy in labor should make these machines attractive in a converter shop turning out a large tonnage. One or two men can charge a cupola by machine, where six or eight do it by hand. The introduction of the machines seems to have been retarded by the fact that the best known ones do not charge the cupola as evenly as a careful gang can do by hand, though they do as good work as many a gang in carelessly run shops. The author has studied the operation of some types of these machines with care, and believes that if they are installed in a proper manner they can be made to operate success-

fully and that they will pay for themselves in a converter shop making say 15 to 20 blows a day or over.

Handling the Cupola.—A word as to the handling of the cupola in Bessemer foundry practice will not be amiss. No one who has not assisted in turning out a long day's run, heat after heat, in a Bessemer foundry, quite realizes the prime importance of hot cupola metal, tapped exactly when it is wanted. The efforts of the metallurgist and blower are set at naught, the advantages of carefully adjusted composition of the vessel charge are lost, and heat after heat of cold steel turned out, when the cupola is badly run and throws down cold metal.

The first factor in good cupola practice is ample tuyère area. In extreme cases, this is made as high as one-third the cross-sectional area of the cupola, and the tuyères are practically a continuous ring. In the author's opinion this is really too great an area, and in fact quite a part of it is soon blanked off by slag accumulating in front of the tuyères during the run. Cupolas are in successful operation in which the tuyère area is as low as 9 or 10 per cent. of that of the cross-section of the cupola. Average practice is probably between these two extremes.

It is absolutely necessary to have ample blower capacity, and the blowers should be able to deliver considerably more than the quantity of air generally prescribed. Usually 30,000 cu. ft. of air per ton of iron melted is considered the most desirable quantity. Cupolas lined to 48 in., melting 8 tons of iron per hour, according to this figure, should need 4000 cu. ft. of air per minute; but in the author's experience, about 4500 cu. ft. per minute should be used most of the time, and occasionally 6000 cu. ft. per minute is blown into these cupolas to hurry them up.

Though many cupolas are provided with fans to furnish the blast, most designers to-day prefer the positive pressure blower. The reason for this preference is that if the tuyères of the cupola get partly plugged up, so that additional resistance is offered to the passage of the air, the fan does not maintain the proper quantity of air per minute; indeed, if the tuyères should be shut tight, no air would flow, and the load on the fan would be a minimum, the air simply going around and around in the casing of the fan. With a pressure blower, on the other hand, a given number of revolutions delivers practically a fixed quantity of air, and up to the strength of the blast pipes and blower, this air simply has to go through the tuyères. It is usually possible to poke out tuyères when they get

plugged, unless iron gets into them, but anyone who has had much experience with a cupola knows that there are times when the available tuyère area becomes very small, and it is good to know that the proper quantity of air is going into the cupola.

The coke should be lighted evenly all over and well burned up, before the entire "bed" is charged. Most authorities say that it is the best practice to fill the cupola up to about 2 ft. over the tuyères with coke, after the fire is well started, and before the first iron is charged. Sometimes, however, with poor or light coke, it is preferable to increase this to 3 ft.

The practice of making the first charge of iron heavier than subsequent charges has the sanction of long use, but in the best shops it is no longer followed, and the charges are all of the same size. Small charges, each followed by a layer of coke, are to be preferred to heavier ones, because when the charges are small, the melting zone is maintained at a more constant level, and the melting conditions are more uniform. The coke used between charges should be about 10 per cent, of the weight of the metal charge, the exact amount depending upon the nature of the coke, percentage of scrap used, and other factors. With the coke is charged enough limestone to make a fluid slag. The function of the limestone, of course, is to unite with and "flux" the ash of the coke, and the silica, and oxides of manganese and iron produced by the oxidation of some of the silicon, manganese and iron of the metal. In the 48-in. cupolas now under the author's supervision, 3000 lb. of metal are used per charge, and from 300 to 400 lb. of coke and six or seven shovels of limestone are used between charges.

Many operators will assure one that they run on a 10 to 1 "coke ratio," that is, that they melt 10 lb. of iron for each pound of coke burned. It will be found, however, that if the total weight of coke (including bed) burned in a cupola in a month and the iron melted in the same period, are compared, the ratio will be more like 8:1 or 9:1. Higher figures than this are not often obtained in every-day practice.

Last, but by no means least, is the question of proper coke, and this is frequently a very difficult problem to solve. If possible, no coke should be accepted that contains over 10 per cent. ash and 1 per cent. sulphur. During the war a great deal of coke had to be used that exceeded these figures, the sulphur sometimes being as high as 1.50 per cent., and even over; as a result, steel was made containing in some cases as much as .12 per cent. sulphur. If the ash content is much over 10 per cent. the value of slag that has to

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be tapped from the cupola is so much increased as to make good running most difficult. The composition of the coke, however, though most important, does not tell the whole story, and unfortunately no tests have yet been devised, except actual trial, that will tell whether a given coke will or will not work well in a cupola.

In general, if the coke is very hard and brittle, it will shatter badly in unloading and handling, so that the pieces are small and the passage of the blast so impeded that the melting rate is seriously decreased. Fines should preferably be taken out by forking the coke from the stock piles to the charging cars, but there is, of course, a limit to the amount we can afford to throw away as fines. Some cokes appear to be of proper hardness and of sufficiently open texture to secure a proper rate of conbustion, and yet do not work well in the cupola. The safest rule appears to be to find a good brand of coke by trial, and then to stick to that brand as long as it continues to be satisfactory.

There is in many foundries a strong prejudice against by-product coke, based apparently mostly upon its appearance, and often not confirmed by actual trial. When by-product coke can be bought that does not exceed 1 per cent. sulphur and 10 or 12 per cent. ash, the author has found it perfectly satisfactory for cupola use, much better in fact than a great deal of the "72-hour" bee-hive coke on the market. During the war, of course, when the coke makers were all at their wits' ends to secure proper coal, thousands of tons both of by-product and of bee-hive coke were sold that were the despair of cupola men, but under normal conditions the author has found by-product coke as uniform and satisfactory as bee-hive.

Most converter foundrymen make the bed of the cupola (from bottom to tuyères) only deep enough to hold about one-fourth or one-third of a vessel charge, and tap the cupola three or four times for each "blow." The object of this practice is to reduce the amount of sulphur absorbed from the coke by the metal; it is thought that if the cupola bed is made deep enough to hold a whole converter charge at one tap, the metal will remain too long in contact with the coke in the bed and absorb more sulphur than if it is tapped as fast as it accumulates. More heat is lost from the metal in the ladle than in the bed of the cupola, but the weight of evidence appears to favor the practice of making frequent small taps.

During the war, several plants, including that with which the author is at present connected, installed apparatus for burning oil in the cupola according to the methods of Bradley Stoughton. In this process, a coke bed of the usual size is used in the cupola, but the amount of coke between metal charges is greatly reduced. An oil burner in each tuyère sends its flame against the incandescent coke bed in the cupola, the usual air blast providing the air necessary for the combustion of the oil, which is atomized by high pressure air. Using the light low-sulphur fuel oils that were on the market at fairly low prices till quite recently, it was possible to obtain lower sulphur in the metal tapped from the cupola than in metal melted with coke alone. This decrease amounted to about .or to .o2 per cent., or for instance from .o8 per cent. S with coke to between .o6 and .o7 per cent. with oil.

It was further claimed that oil melting would prove more economical than the usual practice. This, of course, depends entirely upon the relative prices of coke and of oil, and one shop may show a saving by the use of oil where another does not. In its present stage of development, the author feels that this process for burning oil in the cupola can succeed only where it is possible for the man in charge of the melting shop to give it considerable personal attention, and train his operators most thoroughly in the care and manipulation of the burners. It certainly cannot be put into the hands of the average cupola tender and his helpers with a good chance of success.

Blowing. Bottom-blown Vessel.—With the raw materials commonly at hand in a steel foundry, the best practice for a shop using a bottom-blown converter is to charge heats of pig and scrap that will tap out of the cupola at about 1.60 to 1.80 per cent. silicon, and about .50 to .80 per cent. manganese. In blowing such a heat, the first period or silicon flame lasts from 3 to even 6 minutes, and after some 4 minutes blowing the resistance to the passage of the blast considerably increases and the blast pressure rises, probably owing to the very pasty slag formed. At this stage of the blowing there is practically no flame at the mouth of the vessel, because the oxygen of the blast is all used in the combustion of silicon and manganese, which form slag, and only the inert nitrogen of the air issues from the vessel.

For a bath of given composition, this period of the blow will be the shorter, and the carbon flame will appear the sooner, the hotter the steel is allowed to grow—because as the temperature rises a point is eventually reached where carbon is oxidized in preference to silicon and manganese. Therefore, if the bath is allowed to grow hot undisturbed, carbon may begin to burn before silicon and manganese are

completely eliminated, whereas if the temperature is held down by additions of cold scrap or washed metal, silicon and manganese will be nearly all gone before carbon starts to burn.

When the silicon is nearly eliminated the carbon flame begins to appear, and soon the full long flame, of an intense white color, plays from the vessel, and showers of sparks are thrown out. By the intensity of the light of this flame, the smoke along its edge, the length of the flame and other indications, the blower judges the temperature of the metal, and takes steps to make the blow hotter or colder, according to the methods that are used in the shop. About 8 to 12 minutes after the vessel is turned up, the flame shortens in or "drops, (quite rapidly in the case of this low-manganese mixture), which means that the carbon is practically eliminated, and to prevent the oxidation of iron that begins when the carbon gets low, and increases rapidly as the carbon drops to the final amount of about .08 to .10 per cent., the vessel should be turned down at once, and the blast shut off.

The recarburizers are now added. If used solid, they are preferably thrown into the vessel and a proper time allowed for them to melt, though they may be thrown into the steel as it runs from the vessel. When melted additions are used, they are sometimes poured into the vessel, though often they are poured into the large ladle and the blown metal then run in from the vessel.

The engineer of the blowing engine keeps the pressure up to the full amount of about 25 lb., unless he gets the signal from the blower to slack off. When the vessel is turned down, either at the end of the blow or because anything goes wrong, the engineer at once slows his engine, and stops it when the vessel reaches the horizontal, to avoid blowing out slag and steel. It is for this reason that the engine must be placed where the engineer can see the vessel.

Before taking up the various methods in use for heating or cooling the bath, a few words must be said about the supposed dangers of allowing the metal to become too hot, either before or during the carbon flame period. The author has never heard much about the dangers of overheating the metal in the side-blown converter, but as much of the reasoning in the following paragraphs applies to either the bottom or the side-blown vessel, he has thought it well to consider the subject at some length; for it may be that some of his readers will encounter the same problem in operating side-blown converters that has confronted the author in a shop using the bottom-blown vessel.

The nature of the process is such that there must inevitably be great oxidation of iron at all stages of the blow, since at the mouths of the tuyères the oxygen is in such great excess that not enough silicon, manganese and carbon are at hand to satisfy this oxygen, and some iron must be burned to FeO. But most of this FeO is at once reduced to iron again by carbon, silicon or manganese, as long as there is any considerable amount of these elements present, though some of it unites with the SiO2 and MnO formed and the SiO₂ of the lining, and is lost in the slag. During the first period of the blow nothing comes through the bath (and the rapidly increasing mass of slag) except the inert nitrogen of the air, and probably a little oxygen that has passed through the bath uncombined. When the carbon begins to burn. CO is formed in the metal by the union of carbon and oxygen, escapes in bubbles, and rises rapidly. carried upward by its own heat and by the rush of nitrogen being blown through the vessel. Upon reaching the outside air this CO begins to burn to CO₂, with a flame that will be hotter or colder, more or less luminous, according to the temperature of the metal in the converter; because the hotter the metal, the hotter the gas that is forced through it and the hotter the CO formed in it. The flame itself, of course, is formed at the mouth of the vessel, exactly as a flame is formed when we apply a match to a mass of combustible gas escaping from a pipe.

As the process proceeds, there are fewer and fewer carbon, silicon and manganese molecules to meet the molecules of FeO formed near the tuyères, rob them of their oxygen and return the iron to the bath. Hence, the formation of FeO increases rapidly toward the last of the blow, and when the vessel is turned down there is a considerable amount of it in the metal, besides the far larger part that has found its way into the slag when it met slag particles brought in contact with it by the violent bubbling of the bath. Shimer and Kichline's results¹ show that the amount of oxygen in solution even in over-blown steel is only about .o6 per cent., but for so injurious an impurity this figure is probably larger than it looks. Since the formation of this FeO cannot be prevented, the only question to be considered in this connection is whether the attainment of extreme temperatures increases the formation of FeO, and its retention by the steel.

This question probably must be answered in the affirmative, since it is known that the solubility of FeO in iron increases with increasing purity of the iron, and with increasing temperature. It is

¹ Proc. Am. Inst. Min. & Met. Eng., vol. xlvii (1913), p. 436.

theoretically possible, even, that such extreme temperatures can be reached at the first part of the blow that the iron will dissolve FeO and hold it so tenaciously that even the carbon, silicon and manganese will not reduce it. In practice this is probably never true to any appreciable extent; but in the later stages of the process there is little doubt that with great increase of temperature the power of iron to dissolve its own oxide becomes so great that the affinity of carbon for oxygen is insufficient to reduce the FeO as rapidly as at lower temperatures, and therefore more FeO is formed and held dissolved in the metal.

Moreover because the solubility of the FeO in iron rises with rising temperature, in very hot heats the reduction of this FeO by the manganese of the recarburizers is more difficult. If, therefore, from the exigencies of our business we have to make our metal extremely hot in order to pour very small castings, we face a difficulty due to the greater retention of FeO by our steel, which is then probably more prone to chill rapidly and certainly is more apt to form blowholes in the castings.

This is apparently even more true if we are obliged to blow the metal down to about .10 per cent. carbon, as is usually the case, in order to make castings of about .20 per cent. carbon. It appears to be a fact that "hot" heats in a bottom-blown converter if blown down too low in carbon, have a strong tendency to wildness in the moulds. Such heats require considerable amounts of silicon, aluminum, ferrotitanium, etc., to quiet them, and sometimes even then the metal "rises" in the moulds in the disconcerting fashion already described in another place.

During the war, when manganese was almost a rare metal and every effort to conserve it was imperative, the author had an experience with "wild" heats of steel that was extremely interesting and sheds some light on this question of over-oxidation. For several weeks, we attempted to blow cupola metal containing about 1.50 to 2 per cent. silicon, and very little manganese, seldom over .30 per cent., and often as low as .10 per cent. By increasing the tuyère area of the vessel it was possible to blow this metal quite hot, but great difficulty was experienced in making the steel quiet in the moulds, and this in spite of the fact that the manganese in the finished steel was purposely made quite high, in many cases over 1 per cent. Moreover, test bars from such heats as were poured without undue trouble from wildness, proved that the metal was of very poor quality. The strength and clastic limit were about normal, but the

toughness as shown by extension, contraction of area and bend was very poor, and heat treatment proved entirely inadequate to remedy the trouble; the steel appeared to be "no good" from the start. The difficulty was largely done away with by increasing the manganese in the iron tapped from the cupola; with cupola metal of about 1 per cent. manganese, and 1.50 per cent. silicon, much less trouble was experienced with wild heats, and the number of tests failing to meet requirements was minimized.

Apparently the presence of a certain amount of manganese in the metal charged to the vessel is necessary to prevent the over-oxidation of the bath; and if the steel, especially in the case of very hot blows, is blown just a little too long when there are no metalloids left to protect it, over-oxidation occurs to a sufficient extent to make the metal "wild."

We have found also that if we throw in ferrosilicon too rapidly, in an effort to warm up the metal, we are almost sure to have a wild heat. The reason of this is not clear, though the symptoms are unmistakable; the flame shortens very considerably after the ferrosilicon is added, for the reason of course, that the silicon is being burned instead of the carbon. Why this should result in an overoxidized heat is not clear, unless it is because the flame never regains its proper length, so that the drop is hard to see and the heat is consequently over-blown.

To sum up, the lower the manganese in the metal charged to the vessel, and the hotter the steel is blown, the more care must be taken to turn the heat down before over-blowing begins, and the steel should not be blown any hotter than is necessary to pour the castings.

Cooling the Metal.—The means adopted to cool heats that are too hot are first, to blow steam through the bath with the air by means of a steam pipe (with valve under the blower's control), admitting steam to the wind box. Second, to use cold scrap, pig iron or washed metal, thrown into the vessel before turning up, when it is thought the heat will be too hot, or shoveled in during the blowing. Third, to allow the heat to stand after blowing. The use of steam is not advisable, for though information is not in existence to show positively that it injures the steel, yet the concensus of leading authorities is that the steel is better when no steam is used. Scrap does the steel no harm, as its only effect is to cool the metal because it is itself cold and therefore in warming up and melting it reduces the temperature of the bath. It is best to get it in before the heat is turned up. If one or two heats have blown very hot, and the next

one also runs smoking hot from the cupola, and must be blown at once without cooling off in the receiving ladle, from 50 to 300 lb. of cold scrap in a 3-ton heat will reduce the temperature of the steel considerably. Of course good scrap should be used. Heads or short pieces of gates from the foundry are best, as they can be dropped into the vessel easily. Should the heat blow too hot in spite of the scrap added, or a heat unexpectedly blow very hot, scrap can be dropped into the vessel from a long suspended shovel provided for the purpose.

Because of its purity, washed metal is sometimes used for this purpose, but it is a great waste of money to use it, as it is no more efficient a "cold metal" than scrap. It is no colder than scrap, and therefore absorbs no more heat in warming up; and as its melting point is lower than that of scrap it probably is really less efficient as a cooler, because it melts at a lower temperature and hence takes less heat from the bath in warming up and melting.

Probably the easiest way to handle heats that have blown hotter than necessary is simply to allow them to stand in the vessel or ladle until their temperature has fallen to normal. As the metal cools, the iron retains its oxide less tenaciously, and the oxide is reduced by the carbon and manganese of the recarburizers; and the extra time allowed for this reaction tends to make it more complete.

Warming up the Metal.—A heat whose composition is already fixed may be warmed up in a number of ways. By shoveling 30 or 50 per cent. ferrosilicon into the vessel in lumps, or in pasteboard cans full of fines, silicon is added to the bath and the temperature greatly raised by the heat produced in oxidizing this extra silicon to SiO₂. Generally, this material is added during the carbon flame period, when the blower is able to judge by the appearance of the flame, and the behavior of previous heats, whether or not his steel needs warming up. As stated in the preceding paragraphs, it is necessary to use judgment in adding ferrosilicon during the carbon-flame period, as if it is shoveled in too fast, it interferes with the oxidation of the carbon, with the result of shortening in the flame and it is then harder to judge the proper moment to turn down the vessel.

The use of ferrosilicon for this purpose is mentioned in a patent on the use of manganese as a vessel addition, granted to R. A. Hadfield, dated 1903; and is the subject of a later patent issued to

¹ U. S. Patent No. 743715, dated November 10, 1903.

A. Tropenas¹ in 1908. The Tropenas patent calls for the addition of the ferrosilicon when the carbon flame appears. Hadfield does not specify the exact stage when it should be added. In the Walrand process it was customary to add ferrosilicon quite early in the blow. This process, the invention of Walrand and Légenisel,² which has been used more abroad than in America, is to blow nearly to the "drop of the flame" without additions of any kind; pour into the vessel a certain amount of molten ferrosilicon, containing generally 10 per cent. silicon; turn up and blow until this silicon is eliminated; then turn down and proceed as usual. The iron is protected from oxidation during this "after blow" by the silicon, so that extra loss by oxidation is slight or non-existent, and very hot steel is produced.

Particulars of this method are given in two papers by Mr. G. J. Snelus, in the Journal of the Iron and Steel Institute³ (of Great Britain). In the first article, it is stated that blowing small heats containing a high percentage of silicon, or adding ferrosilicon to the blow during the carbon flame period, had been tried prior to Walrand's invention and had proved a failure, because the gases evolved from the burning carbon carried away much of the heat produced by the combustion of the silicon. Mr. Snelus attributes the success of the Walrand method to the fact that the heat of oxidation of the silicon is rapidly evolved, and that there are no gases (except of course nitrogen), to carry away that heat.

In the second article the statement is made that in an English shop using this method, the vessel was turned down for the addition of ferrosilicon when the bath contained about 1.30 per cent. carbon. The object of turning down before the complete elimination of carbon was to secure the presence of carbon during the after-blow to provide a flame, that there might be a "drop" to show when the added silicon was eliminated from the bath. In most of the shops using the method, the "drop" was determined by the vanishing of the carbon lines in the spectrum of the flame.

In the big steel mills, the method generally followed to warm up a cold heat is to turn the vessel slightly forward so that one or more of the tuyères are at or above the surface of the bath, and blow awhile in this position, slacking the blast to avoid blowing metal

¹ U. S. Patent No. 354694, dated February 25, 1908.

² U. S. Patent No. 503816, dated August 22, 1893, reissue No. 11570, dated 1805.

³ Vol. 1, 1894, p. 26, and vol. 2, 1896, p. 104.

and slag all over the shop. By this means CO is burned to CO₂, within the vessel, by the free oxygen from the exposed tuyère openings; and iron is oxidized to FeO and goes into the slag in considerably increased amounts, resulting in a further gain in temperature. This procedure results in decreased output, owing to greater loss of iron, yet it is cheaper to burn iron than to burn ferrosilicon. If the shop were so arranged as to allow it, therefore, the method would undoubtedly be economical for a small plant. Showers of slag are thrown from the vessel while it is inclined, so that it is not possible to follow this procedure unless there is ample room in front of the converter with no walls, ladles or the like to suffer from the deluge of slag.

Effect of Varying the Silicon and Manganese.—To increase the temperature of the steel by changing the composition of the metal charged to the vessel is the remedy for persistent cold blows, and is the method to be followed as far as possible. To add heat-producing elements is of course the only way to secure additional heat, and this can be done without extra cost by varying the silicon and manganese in the metal charged.

The heat of combustion of silicon is much greater than that of manganese, while that of carbon (to CO) is inconsiderable. At first sight, therefore, it would seem a simple matter merely to increase the percentage of silicon in the vessel charge and thereby make the steel blow very much hotter. But with mixtures containing some 2 per cent. silicon and .60 to 1.00 per cent. manganese, the first period of the blow is very long, sometimes 6 minutes, without any shortening of the carbon flame period, so that a blow requires from 12 to 14 minutes. The pressure of the blast during the silicon-burning period becomes very great and the engine works hard and slows down. The steel produced is not as hot as would be expected. This result agrees with the statement of Walrand and Légenisel that it is difficult to secure hot steel in the bottom-blown vessel by the use of mixtures containing high percentages of silicon.

By the use of mixtures containing rather less silicon than in the above case, and in addition considerable amounts of manganese (the manganese often considerably exceeding the silicon), very good results have been attained. The first period of the blow lasts but r or 2 minutes, the carbon flame comes up very soon after blowing begins, and the total time of a blow is from 8 to 10 minutes. The metal is much hotter than that obtained from the high silicon mixture.

Complete data are not at hand to calculate the total amount of heat contributed to the charge by each of the above mixtures. From a rough estimate, however, it seems that the high-silicon charge contributes more total heat than the high manganese metal. The latter appears to require somewhat less air to burn the metalloids than the former, but not enough to account for the greatly diminished blowing time; and a smaller proportion of the total heat produced would appear to be carried away in the gases, but not enough to account for the hotter steel produced, if the gases in each case are assumed to be heated to the same temperature.

In blowing the high-silicon mixture, evidently no carbon is oxidized till silicon and manganese are largely eliminated, while in the case of the high manganese mixture the temperature at which carbon is oxidized in preference to manganese is soon reached, so that the burning of carbon begins while considerable amounts of manganese (and probably some silicon) remain in the metal. This speeds up the blow, because when carbon is being burned to CO, the bubbling produced by the escaping gas causes the bath to froth up in the vessel and offer greatly decreased resistance to the passage of the blast. That this is so is shown by the lowering of the pressure of the blast in a high silicon heat when carbon begins to burn, with accompanying speeding up of the engine; and by the fact that with the high manganese mixtures the pressure remains low from the first.

Thus the amount of air passing through the bath per minute is increased and the operation shortened. The difference in the two cases, in the amount of heat lost by radiation, and in heating the entering air and the departing nitrogen and CO, is difficult of calculation; but that the air passing in greater volume and at greater speed through the freely boiling high manganese bath has less opportunity to absorb heat from the metal, and hence carries away much less heat, volume for volume, is quite probable. Thus the loss of heat to the gases in the case of the high manganese charge is probably much less than that in the high silicon charge, partly because less air in all is blown through the metal, and partly because the escaping gases are not heated so hot.

At the same time radiation losses are decreased, since the total time of the blow is so much shortened, and the radiation per minute is probably constant; and thus the mixture which gives probably less total heat actually produces much hotter steel.

In addition to their effect upon the heat of the steel, these high manganese mixtures cause marked changes in the slag, because the great amount of MnO produced makes the slag extremely fluid and watery. This is in some respects desirable, because a fluid slag of low melting point doubtless separates more completely from the metal in the ladle than the very gummy slags made with low manganese mixtures. This is because the fluid slags more easily form large globules by the union of small droplets, and these large globules float out of the steel far more rapidly than very finely divided particles. Moreover, the highly manganiferous irons do not pick up so much sulphur in the cupola as lower manganese irons, and if the cupola metal is allowed to stand 10 or 15 minutes in the receiving ladle, some of the sulphur floats out as sulphide of manganese.

The disadvantage of the high manganese mixtures are: first, that a slag rich in MnO corrodes the lining of the vessel severely, and second, that the fluidity of the slag sometimes results in explosive "slopping" of the vessel. This is in part due to the reaction between FeO of the slag and carbon of the steel. The first difficulty is much more than offset by the advantages secured; the second sometimes becomes a great nuisance. The easiest way to stop slopping appears to be to throw a few pounds of 50 or 60 per cent. ferrosilicon into the vessel. This probably acts by temporarily reducing the proportion of MnO and FeO and increasing the SiO₂ in the slag, and thereby rendering it less fluid and active.

The easiest and cheapest way to deal with this slopping is to allow plenty of room in the top of the vessel to take care of the bubbling slag, and to slack down the air pressure by slowing the engine when the vessel gives signs of trouble. When the vessel is newly lined, extra space can be secured by cutting down the weight of the heats, and making up for it by blowing one or two more heats a day; and as soon as the lining is cut away by the slags, there will be room enough to blow full-sized heats. This method has been found to work very well in practice, and the author has seen as many as twenty or twenty-five heats of high manganese metal blown in one day without the use of any vessel additions whatever to prevent slopping.

Published details of German Bessemer practice suggest the use of vessel mixtures running very high in silicon, even up to 3 per cent., with manganese around r per cent. The possibilities of this method have not to the author's knowledge been investigated by the users of small bottom-blown vessels in America. The experience with high silicon irons already mentioned would appear to show that the blowing time would be very long with such mixtures, and the gain in heat not as great as could be desired.

Blowing. Side-blown Vessels.—The metal generally charged to the side-blown vessel contains from 1.75 to 2.00 per cent. silicon, and .30 to .50 per cent. manganese. The vessel is tipped forward and the molten pig iron charged in the proper amount. The weight of the charge varies somewhat from day to day, as the lining of the vessel cuts away, and is fixed by the fact that the level of the metal must be such that it is just even with the bottom edge of the tuyères, when the vessel is tilted forward at an angle of some 7 or 8 degrees. The converter is tipped back to this angle after charging by means of a hand wheel, until the metal reaches the tuyères. If too little metal has been charged, so that the vessel is not inclined forward enough when the metal is at the tuyère level, more is charged till the angle is about right. The blast should strike well out toward the middle of the converter, so that it is important to have this angle correct.

When the correct amount of metal has been charged, the vessel is tipped forward r or 2 degrees, and the blast is turned on, at a pressure preferably of about 3\frac{3}{4} or 4 lb. At first, only sparks come from the mouth of the vessel; and as the metalloids are eliminated from the bath and the volume of the metal decreased, the vessel is gradually turned toward the vertical, so that the blast shall continue to impinge upon the bath at the correct spot. After about 3 or 4 minutes blowing, a faint flame appears at the mouth of the vessel, which soon increases in size; on thus "getting a light," as it is called, the vessel may be tipped back a little, which will assist in bringing up the full flame, and the blast may be somewhat reduced.

The flame at this stage of the blow is short and rather thin, looking much like the ordinary Bessemer flame when it first appears. After some 5 to 15 minutes blowing (the vessel meanwhile being gradually turned nearer and nearer the vertical), the flame begins to lengthen out and grow brighter, and a "boil" takes place. This is accompanied by a flame much like a full ordinary Bessemer flame, and often by the throwing out of a good deal of slag and frequently of metal. In order to keep the charge in the vessel during a boil, the blast is slacked down as low as possible, without filling the tuyères. It is not possible to lower the blast below about r_4^1 lb., without allowing the metal or slag to run into the wind box. The long flame of the boil seldom lasts much over a minute or possibly two, and the flame quickly goes back to its former dimensions. Frequently, there are several boils, at intervals of from 3 to 10 minutes, not all of which are well marked. As many as four or five may occur.

Should the volume of the metal become so low that the blast cannot be made to strike the bath at the proper point, so that the flame goes out, pig iron or ferrosilicon is charged cold, or the vessel is turned forward, the blast taken off, and more fluid metal charged to make the volume right.

It is important in blowing to keep tipping the vessel up so that the blast shall impinge upon the bath at the correct point, and the tuyères be the proper distance above the surface of the metal. If the tuyeres get too high above the metal, the tubes of slag that form at the noses of the tuyères by the chilling of the slag will get so long as to interfere seriously with the blast, partially shutting it off and allowing it to strike at improper points. The result is frequently great oxidation of the metal, and violent slopping.

After one or more boils, followed by the subsidence of the flame to its former size, the "final flame" appears. This is long, intensely white, and feathery at its upper edge, and is the only long flame the vessel gives that lasts more than about a minute. Frequently, slag or metal is thrown out during this stage of the process, but the experienced blower seldom fails to recognize the final flame. The "drop" is much like that of the regular Bessemer flame, and when it appears the vessel is turned down and the blast shut off. The total time of a blow will be from 14 to 25 or 30 minutes.

In the description of bottom-blown practice, attention is drawn to the danger of over-oxidizing the steel, with resulting "wild" heats. With the side-blown converter, this danger is apparently much minimized; indeed the reason the author has reversed his judgment since writing the first edition of this book as to the relative merits of the two styles of converter for small carbon-steel castings is that his observation has convinced him that with no better handling, the side-blown converter is much less apt to turn out over-oxidized and wild heats; it appears to be more "fool-proof."

Nevertheless, many experienced operators like to turn their blows down a bit "young," leaving in the metal part of the carbon they desire in the finished steel, rather than to blow to a well-marked "drop of the flame" and add more carbon in the recarburizers; and this in spite of the fact that it requires more skill to follow this practice, and avoid turning too young, which of course would result in carbon higher than is wanted. The reason for following this procedure is, as in bottom-blown practice, that if the heats are full blown there is danger of over-oxidizing the bath, and making the steel wild in the moulds.

If the blower is uncertain whether or not he has had a final flame and drop, the vessel is turned down and the bath inspected. If the final flame has occurred and the carbon is eliminated, the metal and slag lie quiet and flat in the vessel without bubbling; whereas if there is still carbon in the metal, the slag boils fairly strongly.

It will be seen from the foregoing that the burning of silicon, manganese, and especially carbon, progressess slowly in the side-blown vessel, with one or more periods of sudden increase in the rate of elimination, accompanied by strong boiling of the slag and consequent slopping. Evidently as the blast rushes over the surface of the bath, silicon, manganese, carbon and iron are oxidized, and a slag formed which grows progressively richer and richer in FeO as the carbon, silicon and manganese are eliminated from the surface metal, leaving the iron unprotected and hence more free to oxidize. The circulation of the bath is not sufficient to reduce this FeO by the action of carbon, silicon and manganese to anything like the same extent as is possible in the bottom-blown vessel, where FeO is formed at the bottom of the bath and is largely reduced by the metalloids before it can swim to the surface and enter the slag.

As the oxidation of the metalloids progresses, a point is reached where the slag becomes very high in FeO available for reaction with carbon, and at the same time the fluidity of the slag reaches a maximum. Doubtless the action of the blast is largely confined to the upper layers of the metal, reducing its contents of carbon, silicon and manganese to a very low point, and oxidizing much of its iron to FeO, especially as the surface soon grows so hot and the carbon so low that the oxidation of iron is very rapid. Then the effect of the slag on the metal begins to be more violent, probably assisted by diffusion of carbon into the upper layer from below; the metal and slag begin to boil, the FeO oxidizing silicon, manganese and carbon from the metal rapidly, and soon the boiling becomes very violent. A long flame is thrown out because the CO is evolved so rapidly that it burns to CO2 largely at the vessel's mouth. Upon the exhaustion of the available FeO of the slag this reaction subsides, and the bath, well mixed by the strong boil, is again subjected to slow oxidation, followed by renewed boilings when the slag reaches the proper composition. Apparently, then, quite a part of the oxidation of the metalloids in the side-blown converter is accomplished by the reaction with them of the FeO of the slag, rather than by direct oxidation by atmospheric oxygen, as in the bottom-blown It is perhaps partly for this reason, and partly because the vessel.

metal has far less opportunity to absorb nitrogen and carbonmonoxide, that the side-blown converter is less apt to produce wild steel than the bottom-blown vessel.

If vessel mixtures are blown that contain from .75 per cent. to 1 per cent. manganese with the usual silicon, the boiling and slopping are very violent and result in a great loss of metal. This is no doubt due to the fact that the MnO formed by the combustion of the manganese makes the slag very watery and promotes the reaction between the carbon of the bath and the FeO of the slag.

Ferrosilicon is thrown into the vessel to discourage boiling, as well as to provide extra heat. It probably acts on the slag in the same way as in the bottom-blown vessel, by increasing the SiO₂ and reducing the FeO, thereby rendering the slag less fluid, and postponing its reaction with the carbon of the bath.

So far as the author is aware, the addition of pig iron, washed metal, or scrap in order to cool the steel when it gets too hot is not often resorted to in side-blown practice. The same additions can be used as for a bottom-blown vessel when it is desired to keep the temperature down, as for instance, for a large casting; but as in bottom-blown practice it is probably the best and most economical practice simply to let a hot heat cool off, and if necessary increase the manganese additions to more thoroughly reduce FeO from the steel.

Within the past year, the author has witnessed the operation of a shop where in order to increase the proportion of scrap used, the silicon in the vessel charge was for several weeks kept as low as 1.25 per cent. The practice in this shop was much like that used in the bottom-blown vessels under the author's charge; soon after the flame was well started, 50 per cent. ferrosilicon was thrown into the vessel at intervals. Though the silicon in the metal charged to the vessel was too low to produce sufficient heat, the ferrosilicon provided heat enough to make the metal very hot, and no difficulty was experienced from cold blows.

On a purely experimental basis, the author once had some heats blown under the supervision of A. H. Jameson, in which the silicon was a little under 2 per cent., and the manganese over 4 per cent. Previous to this experiment, operators of side-blown converters supposed that if 1 per cent. manganese gave them sloppy blows, a greater percentage of manganese would increase the difficulty. These heats, however, were blown in both a Stoughton and a Tropenas vessel without great difficulty from slopping, and very hot

steel was produced that ran extremely low in sulphur. The two heats blown analyzed as follows:

| | C | Si | $\mathbf{M}\mathbf{n}$ | S | P |
|-----|------|------|------------------------|------|-------|
| (r) | . 29 | .21 | .74 | .017 | .076 |
| (2) | . 34 | . 26 | .62 | .019 | . 067 |

Apparently the reason that some I per cent. manganese in the vessel charge promotes slopping is that the MnO resulting from its oxidation replaces iron oxide in the slag so that the slag becomes fluid when it contains a smaller proportion of iron oxide than is required to make the slag liquid when low manganese iron is blown. As soon as the slag becomes freely fluid, it mixes with the underlying bath, and its iron oxide reacts with the carbon of the steel, producing a boil. As the MnO does not take part in this reaction, but remains in the slag, together with the additional MnO produced during the boil, the total bases combined with the silica in the slag are not reduced as low during the boil as when this base is almost wholly iron oxide. Therefore, a smaller amount of iron oxide has to be produced before the slag becomes fluid again, and another boil occurs. Thus more frequent boils should be expected than when very little manganese is present in the metal blown.

The theory which the author wished to test out, and on which is based a patent¹ on blowing these high manganese mixtures in the side-blown vessel, is that if only enough manganese be present, the slag will be freely fluid from the time it is first formed, so that as soon as even a little iron oxide is produced and absorbed by the slag, boiling will tend to occur, and thus the boils will be practically continuous, but much reduced in intensity. The above-mentioned experiments seemed to prove that if enough silicon is present to give a hot-blowing charge, these conditions are attained, for such boiling as did occur during the blowing of these heats was far less violent than in usual side-blown practice.

The effect upon the lining, especially at the tuyères, of the fluid slag produced in this process would probably be severe, yet a lining could no doubt be found that would successfully withstand the corrosive action of the slag. Similar iron is blown in the bottom-blown vessel without too great destruction of linings; the problem would be more difficult to solve in the side-blown vessel, on account of the great wear at the mouths of the tuyères, but it could probably be solved. At present prices of manganese, however, manganese steel

¹ U. S. Patent No. 1249075 of Dec. 4, 1917.

scrap should be re-melted in electric furnaces, so that the manganese steel makers would not be justified in attempting to use this process, and as most low-phosphorus pig iron is very low in manganese, no other steel makers are likely to be interested in it. The matter has been discussed here largely for the light it sheds on the metallurgy of the side-blown converter.

Recarburizing in the vessel is very common practice in side-blown foundries. This is owning partly perhaps to the inviting angle at which the vessel stands when "turned down," with its open throat conveniently disposed to throw or pour the recarburizers down into the metal, unlike the bottom-blown vessel which has to be turned into a horizontal position in order that the metal may clear the tuyères when the blast is shut off.

The size of heats and hence the amount of recarburizer varies from day to day, as has been already explained, according to the condition of the lining of the vessel. Generally it will be sufficiently accurate to estimate the increase or decrease in capacity each day and proportion the recarburizer to be added accordingly; but of course if greater accuracy is required, the first heat of the day can be weighed in the ladle and any error in the estimate corrected.

The weighing of melted recarburizer is to be recommended, though if it is melted in crucibles, weighing before melting is generally sufficiently accurate for small additions. Cupola metal can be measured fairly accurately by volume instead of by weight, by using small ladles of known capacity to take off the desired amount. For heavy additions and rapid working, however, platform scales on which the entire addition is weighed in one ladle are essential.

A bottom-blown vessel easily turns out three heats per hour, so that from 18 to 25 heats can be blown per day, running single turn. This rate can be maintained every day in the week. A side-blown vessel takes so much longer to blow each heat that the production is only about two heats per hour, or 9 to 12 heats per day. In some shops, by using larger blowers and increasing the tuyère area, this production is somewhat increased, and as many as three heats an hour are blown. On account of the necessity for repairing the lining, a single side-blown vessel can rarely produce more than 40 heats per week. Side-blown plants, however, generally use two or three vessels, so that a production of 12 to 15 heats per day, or 72 to 90 per week, can be regularly obtained.

These rates of production can be considerably increased, if the cupola capacity is available to provide metal. As a bottom-blown

vessel seldom requires over 12 minutes to blow a heat, and a sideblown 20 to 25 minutes, it is of course possible by driving to turn out steel at a rate greatly exceeding that given, especially for a short Thus in bottom-blown shops, using a single vessel, when three or four heats are required for a large casting, one heat is charged in the vessel and held until the next heat is tapped from the cupola into the receiving ladle. By the time heat one is blown, heat three is ready to tap from the cupola, again filling the ladle, and heat four is tapped from the cupola soon after blowing starts on heat three; thus four heats are turned out as fast as the vessel can take care of them. Since the third and fourth heats are charged into the vessel almost as soon as tapped, they are so hot when charged as to need less silicon than the first and second heats, especially as the steel for a large casting need not, generally must not, be too hot. The average blowing time is reduced by cutting the silicon and manganese in the later heats, and thus it is possible to blow four heats in about 40 or 45 minutes. When two vessels are used, they can be blown alternately with a single engine, and even more rapid work can be done. In the vessels under the author's supervision, six heats have been blown for manganese steel in I hour and IO minutes.

The methods of pouring Bessemer steel vary somewhat with the size of the castings that are being made, and with the preferences of the man in charge. In some shops the whole heat is poured into a large ladle, and castings poured directly from this, or from "shanks" filled from the large ladles. In others, the shanks are filled directly from the vessel. In the latter case recarburizing in the vessel must be the practice adopted, in the former it may be. Again some shops use a stopper and nozzle in the large ladles, some pour "over the lip." The discussion of the relative advantages of these methods properly belongs in the chapter on moulding and pouring, and is given there in some detail.

THE RAW MATERIALS

The raw materials of the baby Bessemer process for steel foundries are:

- 1. Low phosphorus pig.
- 2. Low phosphorus scrap.
- 3. Ferromanganese, ferrosilicon, etc.
- 4. Heads, gates, and "wastered" castings.

The fuel is coke for the cupolas, oil, hard coal, etc., for physic

furnaces other than cupolas, and coal or oil for the boilers of the blowing engine of the bottom-blown vessel.

The increase of phosphorus and sulphur, resulting from the remelting of scrap, is greater in the Bessemer process than in any other, both because the loss of iron in converting to steel is greater, and because a great deal of sulphur is "picked up" from the coke in the cupolas. In order to make steel within the limits of standard specifications, therefore, the greatest care must be taken in the selection of raw materials. In order to keep the phosphorus and sulphur content of the steel below .05 per cent., both pig iron and scrap must contain well under .03 per cent. of those impurities. There is a very limited supply of "low phosphorus" iron, and the price is necessarily high, consequently the greatest possible amount of low phosphorus steel scrap should be worked into the cupola mixture. This scrap is generally used in the shape of plate trimmings and other light material, and in that form is cheaper than the standard "low phosphorus heavy melting scrap" sold for open-hearth use. For open-hearth furnaces this scrap is so light that a great loss is encountered in melting it, but it is well suited to cupola melting. Heavy steel scrap is very hard to melt in the cupola, and should be used sparingly, if at all.

The furnaces smelting the iron ores of Cornwall, Pa., produce a low phosphorus and sulphur pig iron, that contains from .40 per cent. to about .90 per cent. copper. For many years even slight amounts of copper in steel were considered deleterious, and consequently this pig iron was sold considerably cheaper than standard low phosphorus pig. The researches of late years have shown clearly what has long been known in some shops, viz., that copper below .50 per cent., or even .70 per cent., is not at all harmful to steel if the sulphur be kept low; and the use of this copper-bearing pig has greatly increased. As a result, it now commands a price close to that of standard low-phosphorus iron.

Because the absorption of sulphur from the coke is very considerable, a limit for sulphur in coke has to be made, and none should be used with over 1 per cent. of this impurity. It is just as well to draw the specifications to cover the content of ash at the same time, and refuse coke with ash over 10 or 12 per cent. Good foundry coke will conform to these figures, but the choice of a coke, as has already been mentioned, should be governed also by its texture and hardness. Too hard a coke that shatters badly in handling is in general not a good fuel for cupola melting; no specifications, of course, can be

drawn for physical characteristics, and the best way to find out if a coke is suitable is to try a carload or so of it, always bearing in mind the natural conservatism of the average workman, who has an inveterate habit of getting used to one brand of coke, or one shape of scrap, and moving heaven and earth to prevent the purchase of any material other than what he has been using.

The proportion of purchased scrap that can be used is governed by the amount of heads, gates and "wastered" castings that the shop produces and has to melt up. Roughly from 30 to 50 per cent. of the metal in the ladle finds its way to scrap, so that at least 30 per cent. of the cupola charges has to be shop scrap. This re-melting of scrap causes an increase of the phosphorus and sulphur content of the steel of successive heats, which, as Mr. Jameson has shown us, works up to a limit by progressively decreasing increments.

Taking his equations, let us see what are the limits of phosphorus, sulphur and copper content that we shall attain in our steel, using a mixture of 30 per cent. our own scrap, 10 per cent. purchased scrap and 60 per cent. pig iron, of which two-thirds is iron containing .70 per cent. copper, one-third standard low phosphorus iron. Phosphorus and sulphur in purchased pig and scrap, .03 per cent. Converting loss, 12 per cent. (a good average for bottom-blown vessels). "Pick-up" of sulphur in melting, .01 per cent. Then, for the limits of phosphorus, sulphur and copper reached by our steel, using the formulæ,

$$X = \frac{(100 - a)c}{100 - (a + b)} \text{ and } X = \frac{(100 - a)c + 100d}{100 - (a + b)}$$

Where X = per cent. element in question after maximum is reached.

a = per cent. scrap re-melted = 30 per cent.

100 - a = per cent. new material melted = 70 per cent.

b = converting loss = 12 per cent.

c = per cent. element in question in pig and plate scrap = .03 per cent.

d = "pick up" (.or for S).

For Phosphorus²

$$X = \frac{70(.03)}{58} = .036 \text{ per cent.}$$

¹ See Chapter XII.

² Sometimes phosphorus as well as sulphur may be picked up from the coke in the cupola. When this is the case, the second formula should, of course, be used for this calculation.

FOR SULPHUR

$$X = \frac{70(.03) + 100(.01)}{58} = .053 \text{ per cent.}$$

FOR COPPER

$$\frac{40(.70)}{70} = .40 \text{ per cent. copper in new material}$$

$$X = \frac{70(.40)}{58} = .48 \text{ per cent.}$$

Since the ferromanganese used as recarburizer contains considerable amounts of phosphorus, generally averaging .30 per cent., it is necessary to take this factor into account in figuring out our X or maximum phosphorus that we will attain in our steel. To show clearly the influence of this factor in the problem, the following examples are appended, giving actual calculations made on several steels in which varying amounts of ferromanganese are used, depending on the amount of manganese desired in the finished product. From these figures it will be seen that the ferromanganese has a considerable effect on the phosphorus content of the steel. This effect, of course, increases with the amount of ferromanganese used per heat.

ORDINARY STEEL WITH 1.30 PER CENT. MANGANESE

80 lb. ferromanganese at .3 per cent. phosphorus physic 22 lb. ferrosilicon at .1 per cent. phosphorus

r400 lb. foundry scrap at X per cent. phosphorus 4600 lb. new pig and scrap at .03 per cent. phosphorus charged to cupola 5350 lb. steel produced

$$X = \frac{80(.3) + 22(.1) + 1400X + 4600(.03)}{5350} = .042 \text{ per cent.}$$

ORDINARY STEEL WITH .60 PER CENT. MANGANESE

Using the same mixture, but only 40 lb. ferromanganese for physic.

$$X = \frac{40(.3) + 22(.1) + 1400X + 4600(.03)}{5300} = .039 \text{ per cent.}$$

Figuring Cupola Charges.—To illustrate the methods of figuring cupola charges for Bessemer work, a few typical examples are here

introduced. In this connection it is as well to state definitely that "mixing" has to be done wholly by analysis, and for that reason the use of several "brands" or makes of pig iron is not at all necessary. It will be found advantageous, nevertheless, to use at least four lots of pig iron for mixtures, because it is quite out of the question to obtain pig of uniform silicon content, and in order to keep the composition of the successive heats as nearly constant as possible from day to day, without constant figuring of new charges, several lots of iron must be used. Generally two lots will give out nearly together, and then two more can be selected whose average silicon content is the same as the average of the two replaced. With four lots of pig and two kinds of scrap, composition can be kept fairly even for weeks by judicious selection, without the necessity of refiguring mixtures.

Desired Si 2.10 per cent.

| Kind | Si, per cent. | Mn, per cent. | Cu, per cent. | Wght. | Si, lb. | Mn, lb. |
|----------------------|------------------|---------------------|---------------------|-------|---------|---------|
| Pig 1 Std. low P | 3.00 | . 30 | 0 | 1000 | 30.0 | |
| Pig 2 Std. low P | 2.20 | . 30 | 0 | 800 | 17.6 | |
| Pig 3 Cornwall low P | 3.30 | . 30 | .60 | 1400 | 46.2 | 12.6 |
| Pig 4 Cornwall low P | 2.50 | .30 | .50 | 1000 | 25.0 | |
| Foundry scrap | .40 | .60 | .65 | 1800 | 7.2 | 10.8 |
| | | | | | | |
| | | | | 6000 | 126.0 | 23.4 |
| | | | | | 2.10% | .39% |

Desired { Si 2.00 per cent. Mn 1.50 per cent.

| Kind | Si, per cent. | Mn, per cent. | Cu, per cent. | Wght. | Si, lb. | Mn, lb. |
|------------------|------------------|---------------------|---------------------|-------|---------|---------------|
| Pig 1 Std. low P | 3.00 | 1.80 | 0 | 1000 | 30.0 | 18.0 |
| Pig 2 Std. low P | 2.20 | 2.20 | 0 | 1100 | 24.2 | 24.2 |
| Pig 3 Std. low P | 3.30 | 1.60 | 0 | 900 | 29.7 | 14.4 |
| Pig 4 Std. low P | 2.50 | 1.90 | 0 | 1 200 | 30.0 | 22.8 |
| Foundry scrap | .40 | .60 | .65 | 1800 | 7.2 | 10.8 |
| | | | | 6000 | 121.1 | 90.2 1.50% |

| Desired | Si 1.80 per cent. Mn low |
|---------|-----------------------------|
| Desired | Mn low |

| Kind | Si, per cent. | Mn, per cent. | Cu, per cent. | Wght. | Si, lb. | Mn, lb. |
|---------------------|------------------|---------------------|---------------------|-------|--------------|--------------|
| Pig 1 Std. low P | 2.00 | . 30 | | 700 | 14.0 | 2 . I |
| Pig 2 Std. low P | | 1.50 | | 800 | 20.0 | 12.0 |
| Pig 3 Lebanon low P | 2.90 | .30 | .60 | 1000 | 29. 0 | 3.0 |
| Pig 4 Lebanon low P | 3.40 | . 30 | . 50 | 1100 | 37 · 4 | 3 · 3 |
| Foundry scrap | .40 | .60 | .65 | 1800 | 7.2 | 10.8 |
| Plate scrap | . 10 | .60 | | 600 | 0.6 | 3.6 |
| | | | | 6000 | 108.2 | 34.8 .58% |

This work is so much a matter of arithmetical juggling that it is not worth while to give more than these few examples. Low phosphorus scrap can be used in greater or less amounts to balance variations in silicon in the pig iron. If there is a shortage of foundry scrap, plate scrap should be worked in to take its place and save pig iron, the most costly raw material used. If much scrap is to be melted (it is generally best to melt at least 40 per cent.), the pig irons bought have to be correspondingly high in silicon; and the man in charge of this work has to keep close watch of his stocks of raw material in order to specify the silicon content of his pig iron to suit his future requirements. The use of 10 per cent, ferrosilicon and steel scrap to take the place of part of the pig iron is perfectly possible, though it is hard to make the scrap and ferrosilicon melt at a sufficiently even rate in the cupola to give heats of the uniform composition so essential for good blowing. Even in melting pig iron and scrap, the unevenness of composition of successive heats, due to the more easily fusible materials of the charges melting first and running ahead of the other stock, sometimes gives the blower trouble. A large proportion of plate scrap can be melted in the cupola charge, and the shortage of silicon made up by the use of 10 per cent. ferrosilicon, or possibly even by adding 50 per cent. ferrosilicon melted separately. Generally there is no financial advantage in using these materials, as it is cheaper to use pig iron. As mentioned in the introductory chapters, in places where pig iron is costly and scrap cheap, the use of these materials in large percentages may be advantageous.

The silicon and manganese in the metal run from the cupola will be somewhat lower than in the metal charged, on account of oxidation in melting, some 20 per cent. of the silicon being so lost. Of the two, the manganese suffers the greater loss, when it is present in considerable amounts. The exact composition of the metal tapped will vary somewhat from heat to heat.

When a given charge is used, the silicon and even more the manganese content of the metal run from the cupola tend to fall off, toward the end of the day's run. This can be taken care of by changing the mixture used for the later heats, but one should be careful not to go too far in this direction, since the cupola metal toward the last of the run is often hotter than that tapped earlier, so that in these heats less manganese and silicon are needed for the production of hot steel.

Figuring Recarburizers.—The figuring of additions to the steel is so simple a matter that a very few words must suffice to illustrate it. The materials available for this purpose are:

| | Material | Source of |
|-----|---------------------------------|-------------|
| I. | Cupola metal | C (Si)(Mn) |
| 2. | 80 per cent. ferromanganese | C Mn |
| 3. | 20 per cent. spiegel | C Mn |
| 4 | 50 to 60 per cent. ferrosilicon | Si |
| 5. | 10 to 12 per cent. ferrosilicon | C Si |
| 6. | Silico-spiegel | C Si Mn |
| 7. | Pig iron | C (Si) (Mn) |
| 8. | Hard coal, coke, etc | C |
| 9. | Silico-manganese | C Si Mn |
| 10. | Nickel | Ni |
| ıı. | Ferrochromes | (C) Cr |

The choice among the many materials that can be used depends largely upon three considerations, as follows:

- T. Are additions to be melted or solid?
- 2. If melted, by what method?
- 3. Is the steel to be of high or of low carbon content?

If solid additions are to be used, naturally the choice falls upon materials as high in the desired elements as can be obtained, unless too high a melting point results from great purity. Eighty per cent. ferromanganese, 50 per cent. ferrosilicon, etc., are then used, and extra carbon, if needed, obtained from additions of coke or of cupola metal.

If much hard steel, or steel high in various alloys, is to be made, the recarburizer generally has to be melted. As far as possible, metal from the main cupolas should be taken to make up recarburizer for hard steel heats, since it saves the melting of large separate lots of recarburizer.

The melting furnaces that are used are generally of a sort suited to intermittent operation, because a Bessemer plant is seldom run continuously. The furnaces that can be used are:

- 1. Small cupola.
- 2. Air furnace, or oil-fired furnace of some kind.
- 3. Crucible furnace, generally oil or coal fired.
- 4. Electric furnace.

The choice is governed by

- 1. Nature of material to be melted.
- 2. Amount of material to be melted.
- 3. Cost of melting.
- 4. Continuous or intermittent operation.

The cupola is suited to melting low percentage ferro alloys, and melts large quantities cheaply. The aurthor has successfully operated a baby cupola, 18 in. inside the lining, for melting silicospiegel and a certain amount of 80 per cent. ferromanganese for recarburizing steel of .25 to .30 per cent. carbon and 1 to 1.30 per cent. manganese. As a rule, however, the higher percentage ferroalloys such as 80 per cent. ferromanganese, when used molten, have until late years been melted in crucibles, or in a small electric furnace. In many cases, the latter will melt the alloys more cheaply than is possible in crucibles. Under certain circumstances, for instance where coke is costly and oil cheap, the air furnace or some type of non-crucible oil melting furnace may be economically employed.

In the manufacture of manganese steel, about one-fifth of the weight of the finished steel is ferromanganese, one part of the latter and four parts of blown metal being mixed in the ladle. Obviously, so large an addition of recarburizer must be added molten. For many years this melting was done in crucibles, but in 1916 the small cupola was successfully applied to this work under the author's direction, and has since been introduced in other shops. Several years before that date, our attention had been drawn to the practice of melting ferromanganese in small movable cupolas, each of which was used for a single charge, then taken from under its stack, the slag drained out, the fire re-made, etc., before being charged again. Various precautions against excessive loss by oxidation in these cupolas were recommended; but the installation was too cumbrous for our needs.

After considerable experimenting, we found that by observing a

few simple precautions an ordinary fixed cupola could be made to melt ferromanganese satisfactorily, with an average melting loss (by weight) of 6 to 9 per cent. The composition of the metal tapped is the same as that of the metal charged.

The cupola we use is lined to 30 in. and is provided with four tuyères, set at such a height as to permit the accumulation of from 1100 to 1200 lb. of metal in the bed. The blast pressure is kept extremely low, as higher pressures lead to increased melting losses.

The flux used is chiefly old clay brick-bats, broken up to about the size of eggs. As the oxidation product from the melting of ferromanganese is almost entirely MnO and FeO, limestone or any other base alone is unsuitable for fluxing, as acid material is needed; the old clay bricks, consisting chiefly of SiO₂ and Al₂O₃, make an ideal flux, and produce with the MnO and a small amount of limestone, a freely flowing slag that is sold at a fair figure for blast furnace use. Unless a suitable flux is used in the cupola, the lining is severely corroded.

Finally, we found that a large proportion of the total loss of ferromanganese occurred when the cupola was nearly empty. By charging a sufficient excess over the day's requirements to have the cupola partly full when the last tap is made, we keep the losses to a minimum. Application has been made for patents on these features, which have been the means of solving a very difficult problem. Early in 1916 the quality of crucibles we were able to buy had become so poor that it was impossible for us to continue using them; but after the cupola had been in operation a few months, as one of our executives said, we tapped half a ton of ferromanganese every 20 minutes "just like getting it out of a pump!"

The loss of 6 to 9 per cent. of the ferromanganese of course adds considerably to the melting cost, but the other expenses are so low that except when ferromanganese is at extremely high prices, the total cost of the cupola melting, even counting the lost metal, is far lower than can be secured by any other process.

Whenever possible, the recarburizers should be used molten, as there is no doubt that they purify the steel far better when so added. When the additions are but a small fraction of the weight of the heat, especially if the steel is hot enough to melt them readily, and time is allowed for them to melt and act on the steel, quite good work can be done with solid recarburizers. But molten metal would do better.

Summarizing this question, the methods used in adding recarburizers are as follows:

- 1. Adding solid in ladle—least effective.
- 2. Adding solid in vessel—better.
- 3. Adding liquid in ladle—better yet.
- 4. Adding liquid in vessel-best.

If low-carbon steel high in silicon and manganese is to be made, the choice must fall on high percentage silicon and manganese alloys because, as the following figures show, the low percentage alloys contain too much carbon per unit of silicon or manganese. For instance, a steel of silicon .40 per cent., manganese .80 per cent., using 20 per cent. spiegel and 10 per cent. ferrosilicon, would figure out as follows:

| Kind | C, per cent. | Si, per cent. | Mn, per cent. | Wght. | C lb. | Si lb. | Mn lb. |
|----------------------|-----------------|---------------|---------------------|--------------|--------------|--------------|--------------|
| Spiegel Ferrosilicon | 4 · 5 | 10 | 20 | 35·4 30.8 | 1.59 0.55 | 3.08 | 7.08 |
| Blown metal | . 10 | .10 | . 10 | 934.0 | .93 | .93 | .93 |
| | | | | 1000.2 | 3.07 | 4.01 .40% | 8.01 .80% |

These alloys thus contain too much carbon for our purposes, if we desire steel below .30 per cent. carbon.

Using 80 per cent. ferromanganese and 50 per cent. ferrosilicon, we figure as follows:

| Kind | C, per cent. | Si, per cent. | Mn, per cent. | Wght. | C lb. | Si lb. | Mn lb. |
|----------------|-----------------|------------------|---------------------|-------|-------|--------|--------|
| Ferromanganese | 6 | | 80 | 8.8 | . 528 | | 7.04 |
| Ferrosilicon | . 5 | 50 | | 6.0 | .030 | 3.0 | |
| Blown metal | .10 | .10 | .10 | 985.0 | . 985 | . 985 | . 985 |
| • | | | | 999.8 | 1.543 | 3.985 | 8.025 |
| | | | | | . 15% | .40% | .80% |

To make steel of .20 per cent. carbon, the extra half pound of carbon needed can be added as coke, in which case we add 1 lb., allowing for a loss of 50 per cent.; or better we use cupola metal or pig iron. If this contains 2 per cent. silicon, .50 per cent. manganese,

| 3.5 per cent. carbon, we have to cut our ferrosilicon and ferroman- | |
|---|--|
| ganese accordingly, thus, | |

| Kind | Wght., lb. | C lb. | Si lb. | Mn lb. |
|-----------------------------|------------|----------|--------|---------|
| 50 per cent. ferrosilicon | 5.5 | .0275 | 2.75 | |
| 80 per cent. ferromanganese | 8.7 | . 5220 | | 6.96 |
| Cupola metal | Ι2 | . 4200 | . 24 | . 06 |
| Blown metal | 975 | .975 | - 975 | .975 |
| | 1001.2 | 1 . 9445 | 3.965 | 7 - 995 |
| | | . 19% | .40% | .80% |

For a steel of this composition, silicomanganese containing some 20 per cent. silicon and 60 to 70 per cent. manganese can now be obtained, which is almost ideal for the purpose. It contains, however, only 1 per cent. carbon or under, so that other alloys usually have to be used with it.

On the other hand, for steels above .34 per cent. carbon or so, the spiegel and ferrosilicon will not contain enough carbon for our purposes, and we must either melt pig iron with them, or use cupola metal to make up the deficiency as before. Whichever we do, allowance must be made for the silicon and manganese in the pig iron or the cupola metal. Even in this case, however, it is not by any means necessary to use the low percentage alloys, melted in the cupola, as it may sometimes be cheaper to use the higher percentage ferros added solid, or even melted. For steel of .45 per cent. carbon, .40 per cent. silicon, .80 per cent. manganese, we have the following four methods open to us.

1. Using 20 per cent. spiegel, 10 per cent. ferrosilicon, and pig iron (carbon 3.5 per cent., silicon 1 per cent., manganese .50 per cent.).

| Kind | Wght., lb. | C lb. | Si lb. | Mn lb. |
|---------------------------|------------|-------|--------|--------|
| 10 per cent. ferrosilicon | 27 | .486 | 2.7 | |
| 20 per cent. spiegel | | 1.575 | | 7.00 |
| Pig | | 1.505 | . 43 | . 215 |
| Blown metal | | .900 | .90 | . 900 |
| | 1005 | 4.466 | 4.03 | 8.115 |

^{2.} Using 20 per cent. spiegel, 10 per cent. ferrosilicon, and cupola metal (3.5 per cent. carbon, 2 per cent. silicon, .50 per cent. manganese).

| Kind | Wght., lb. | C lb. | Si lb. | Mn lb. |
|---------------------------|------------|------------------|--------|--------|
| 10 per cent. ferrosilicon | 23 | .414 | 2.3 | |
| 20 per cent. spiegel | 35 | 1.575 | | 7.00 |
| Cupola metal | 46 | 1.610 | .92 | . 23 |
| Blown metal | 900 | . 900 | . 90 | .90 |
| | 1004 | 4 · 499 · 45% | 4.12 | 8.13 |

3. Using 50 per cent. ferrosilicon, 80 per cent. ferromanganese, and cupola metal.

| Kind | Wght., lb. | C lb. | Si lb. | Mn lb. |
|---------------------------|------------|-------------------------------|--------|----------------------|
| 50 per cent. ferrosilicon | | .015 .510 3.185 .900 | 1.50 | 6.80 -455 .900 |
| | 1002.5 | 4.610 | 4.22 | 8.155 |

4. Using 50 per cent. ferrosilicon, 80 per cent. ferromanganese and pig iron. This calculation will be very much like number 3.

As we have already set forth at some length in the introductory chapters, Bessemer metal is more overoxidized than open-hearth steel, and even when the deoxidizers are added molten in the converter, the removal of oxides is never complete. Nevertheless, if both ferromanganese (80 per cent.) and ferrosilicon (generally 50 per cent.) are added as deoxidizers, and especially if they are used molten, steel can be produced that is comparatively free from oxides and gases, and castings are poured that are quite free from blowholes, etc. The difficulty of deoxidizing the steel sufficiently to prevent blowholes increases greatly as the carbon content of the steel produced is reduced. Thus some shops that add enough ferromanganese, ferrosilicon and pig iron (or metal from the cupola), to produce steel of .50 per cent. carbon with the usual silicon and manganese, pour castings of this metal without great difficulty; but ask them to pour .20 per cent. or .15 per cent. carbon steel and they get into trouble with blowholes and misrun castings.

To examine this question a little more closely, we may take an example from practice. At one shop using a small bottom-blown vessel, it was found that composition (A) (below) would run well, and gave castings with few blowholes; composition (B) soon became

pasty in pouring, ran badly and gave castings with many blowholes; this trouble was corrected by substituting composition (C) for composition B.

| | A | В | C |
|----|---------------|----------------|----------------|
| C | | . 20 per cent. | . 20 per cent. |
| Si | | . 30 per cent. | .30 per cent. |
| Mn | .60 per cent. | .60 per cent. | 1.30 per cent. |

In explanation of this, it is suggested that steel B becomes sticky sooner than A or C, for the following reasons.

- 1. Carbon and manganese both lower the melting point of steel, therefore B's melting point is highest.
- 2. Carbon and manganese both increase the fluidity of molten steel, especially just before freezing. Steels low in carbon, silicon and manganese go through a longer period when they are sticky than those high in one or all of these elements. Therefore B soon became sticky.
- 3. The purer the iron, the more tenaciously it holds its oxides. Therefore B is probably less well cleansed of oxide than A, because it is more nearly pure iron, and therefore the same amount of silicon and manganese less thoroughly removes the oxide; and less well than C because the lower manganese less thoroughly removes the oxide.
- 4. The lower the temperature of the iron, the less tenaciously it holds its oxides, and the greater the tendency of carbon, silicon and manganese to reduce them. Therefore B is probably less well freed of oxide than A or C, because it freezes at a higher temperature.
- 5. Probably oxides both raise the melting point of the steel, and decrease its fluidity, hence B will be less fluid and have a higher melting point than A and C from this cause also. This follows from considerations 3 and 4.

The stickiness of steel of composition B rendered it a difficult matter to run such a steel into small sections. A 3-ton heat soon became too "mushy" to run into light work, and many of the castings poured of metal approaching the mushy state were full of blowholes.

There are many ways in which the greater abundance of blow-holes in steel B than in A or C may be explained, the most probable being that owing to the higher melting point of B, less time is allowed for the escape of gases from this steel, which are therefore carried over in considerable quantity into the mushy stage just before freezing, when their escape is greatly hindered by the resistance of the

mushy steel to their egress. It may be added that additions of aluminum, silicon-calcium and other deoxidizers improved the pouring qualities of steel B, but as the same additions were used for steels A and C, the relative ease of handling was not affected by these additions; and as steel C was found to be superior in physical properties to either A or B, it soon displaced them both in the foundry. It was as strong or stronger than A, and as tough or tougher than B, without being in any way too hard for common use. This foundry had for years had difficulty in pouring steel of as low as .20 per cent. carbon and had been deterred from trying composition C by the opinion, long commonly held, that more than I per cent. manganese rendered the steel brittle. No doubt harder steels, containing from .45 per cent. carbon up, such as are used for the manufacture of rails, are rendered brittle by such high percentages of manganese. At any rate, the rail mill men formerly held that I per cent, was about the maximum content of manganese that it was safe to add to their But in the case of soft steels, much more manganese can be used not only with safety, but even with great benefit to the steel. This is perhaps because the functions of manganese and of carbon in steel are much alike, so that it is the sum of their percentages, or the sum of some multiple of the percentage of each, that cannot exceed a certain figure without producing a brittle steel.

Using coke or coal in the ladle as a source of carbon is a practice to be avoided whenever possible. To sum up this subject, the composition of typical recarburizers and the methods of using them are given in the following table.

| | C, | Si, | Mn. | Ni, | Cr, | | | Melted |
|--------------------------|--------|-------|--------|-------|-------|--------------------------|-----------------|-----------|
| Material | per | per | per | per | per | $\mathbf{U}\mathbf{sed}$ | In | |
| | cent. | cent. | cent. | cent. | cent. | | | in |
| Cupola metal | 3-4 | 1-2 | .5-3 | | | Molten | Vessel or ladle | С |
| Ferromanganesc. | 6-7 | . 30 | 80 | | | Molten, | Vessel or ladle | C-Cr-E |
| | | | | ļ | | solid | | |
| Spiegel | 4-5 | .30 | 20 | | | Molten | Vessel or ladle | C-A |
| Ferrosilicon | 0.5 | 50 | | | | Molten, | Vessel or ladle | Cr-E |
| | | | ļ | | | solid | | |
| Silicomanganese. | 1.0 | 20-25 | 50-70 | | | Molten, | Vessel or ladle | C-Cr-E |
| | | | | | | solid | | |
| Ferrosilicon | 1.8 | 10 | | | | Molten | Vessel or ladle | C-A |
| Silicospiegel | 1.9 | 10 | 20 | | | Molten | Vessel or ladle | C-A |
| Pig iron | 3-4 | 1-3.5 | . 30-3 | | | Molten | Vessel or ladle | C-A |
| Hard coal, coke | 50 | | | | | Solid | Ladle | . |
| Nickel | . 50 | | | 99 | | Molten, | Vessel or ladle | Cr-E |
| | | | | | | solid | | |
| $Ferrochrome \dots . \\$ | 7-8 | | | | 70 | Molten | Vessel or ladle | Cr-E |
| Ferrochrome | . 50-2 | 1 | · | ļ | 70 | Molten | Vessel or ladle | Cr-E |

C = Cupola. Cr = Crucible. E = Electric furnace. A = Air furnace.

ALLOY STEELS

The composition of Bessemer steel may be varied within wide limits with comparative ease, since the blown metal is practically pure iron, containing only about .ro per cent. carbon, silicon and manganese, and therefore by properly proportioning the recarburizers, the chemical content may be made almost anything we please. In this connection, however, it should be remembered that if heavy additions are to be made to the steel, in order to secure high carbon content, or a high content in manganese, nickel, chrome, or other alloy, it is often essential to melt these additions. If this is not done, the steel will be so chilled that it will be impossible to pour it; indeed, in the case of some of the alloy steels now made in great quantities, to attempt to use solid additions would result in practically solidifying the steel in the ladles or vessel.

The limitations of the Bessemer process in the production of alloy steels are chiefly confined to chrome, tungsten and molybdenum steels. These alloys often have to be added molten, and ferrochromes are very hard to melt alone, ferrotungsten and ferromolybdenum even more so, while metallic tungsten and molybdenum cannot be rhelted alone. With sufficient amounts of ferromanganese or pig iron, ferrochromes can be melted, but if crucibles are used for melting, low carbon ferrochromes will absorb carbon from the graphite of the crucibles very rapidly, so that the production of low carbon chrome steels of high chrome content is not at all easy. Tungsten and molybdenum steels of high-alloy content and low carbon are almost impossible to produce by Bessemer methods.

In the manufacture of nickel-chrome steel, the Mayari pig irons can be used to great advantage, as almost all of the nickel remains in the metal after melting and blowing. The chrome is reduced by oxidation losses from about 2 per cent. to some .50 to .70 per cent. Advantage can be taken of the fact that nickel is not oxidized in blowing, to produce nickel steel by charging the cold nickel in the form of shot in the vessel before turning up the heat. The alloy is melted and thoroughly incorporated in the steel during the blow. Similarly, if nickel and chrome, or nickel-chrome steels are manufactured in quantity, it will pay to keep the scrap from these heats separate and use it in the manufacture of the alloy steel, because some of the chrome and nearly all of the nickel can be counted upon to remain in the metal and reduce the consumption of nickel and ferrochrome proportionately.

The Bessemer process is particularly well suited to the manufacture of manganese steel, for the following reasons.

First, Bessemer blown metal and ferromanganese, mixed together in such proportions as to give the manganese content desired, result in a steel of correct composition in carbon and silicon.

Second, manganese steel is poured largely into comparatively small castings; hence the very hot Bessemer metal, made in small heats at frequent intervals, is well suited to the pouring problems presented.

Third, the enormous dose of ferromanganese added to the blown metal deoxidizes it so completely that the quality of the steel is as high as can be made by any process of steel making. To what an extent this is true was bardly realized until the manufacture of manganese steel in the electric furnace was reduced to practice. The superior quality of electric furnace steel is so well proved that it was anticipated that electric furnace manganese steel would be better than the converter product. A very large number of carefully conducted tests, however, has demonstrated beyond question that manganese steel made in the electric furnace is fully equal, but in no way superior, to that produced in the converter. Moreover, because it is so easy to make steel very hot in the electric furnace, occasional heats are produced by that process which are too hot to pour and manganese steel when poured hotter than necessary is not so strong and tough as when cast at the proper temperature.

The special problems presented in the manufacture of this steel are the melting of the ferromanganese, which is used in such great quantities that it cannot be added solid, and the working up of the scrap. The former question has already been discussed at some length.

The scrap problem is a hard nut to crack, because in remelting and blowing this metal the manganese is necessarily all oxidized and lost. Cupola or crucible melting of the scrap for use direct is not feasible, because of the great absorption of carbon that necessarily occurs in the process.

When the bottom-blown converter is used, a considerable proportion of the scrap can be melted in the cupola charge, and with the side-blown converter this may be possible. At present prices of manganese, however, this is a pretty wasteful method of using up the scrap; to sell it to open-hearth steel makers does not help matters much, because they recover but a small part of the manganese, and are usually unwilling to pay as much for it as for plain carbon-steel scrap.

The electric process is the ideal way of working up the scrap, because in the electric furnace it can be remelted with very little loss of manganese, and used again as manganese steel. At present prices of ferromanganese this results in a considerable saving; but the installation of the electric furnace is costly. However, for manganese-steel manufacture, the ideal installation today, in places where pig and coke are obtainable at reasonable figures, is undoubtedly the converter to make the steel from pig iron, steel scrap and ferromanganese, and the electric furnace to remelt the manganese steel scrap. Of course, in places where the converter is at a disadvantage on account of high prices of pig, coke, etc., the electric furnace alone would be the best installation, melting steel scrap and ferromanganese as raw material, and using manganese steel scrap as part of each charge.

The Basic Bessemer Process.—We have seen that in the Bessemer process, as carried on in the United States, no phosphorus or sulphur is eliminated from the steel; in fact, owing to the loss of carbon, silicon, manganese and iron in blowing, without any loss of phosphorus and sulphur, the percentage of these impurities somewhat increases. In the case of sulphur, and to some extent phosphorus, an actual increase in the total weight of the impurity occurs, because some is absorbed from the fuel in the cupola. Because the lining of the vessel, and therefore the slag, is acid, i.e., siliceous, any phosphorus oxidized by the blast to P2O5 is at once reduced by iron and carbon, with the formation of FeO,CO, and metallic phosphorus, since iron and carbon have a greater affinity for oxygen at these high temperatures than has phosphorus; and in the absence of any base such as lime (CaO) to hold the P2O5 in the slag as phosphate of lime, permanent oxidation of the phosphorus is impossible. Phosphate of iron, if formed, will at once be broken up by SiO₂, as SiO₂ has a greater affinity for FeO than has P₂O₅, at these temperatures. Thus no phosphorus can be eliminated in acid-lined vessels.

The basic Bessemer process was designed to overcome this difficulty, and give a rapid process of steel making that should at the same time be a dephosphorizing process. The first requisite being a basic slag, lime is added to the charge in the vessel before blowing begins. Since this lime would at once attack a lining of silica (SiO₂) or any acid material, eating into it until enough SiO₂ was dissolved out to unite with the CaO to form a siliceous slag, the lining of the basic Bessemer vessel has to be of either neutral or basic material. On account of its cheapness, a basic lining of burned

dolomite (double carbonate of lime and magnesium) is used. Though in fact this lining is partly eaten away by each heat and contributes material to the basic slag, this contribution is not necessary, is indeed an evil, for if not thus cut into the linings would last much longer.

In order to keep the amount of lime needed to produce a basic slag, and hence also the volume of slag produced, at a minimum, pig iron low in silicon is blown; in practice the silicon should not be over Since this takes away the greater part of the heat-.50 per cent. producing element utilized in ordinary Bessemer work to warm up the charge, some substitute has to be found, and that substitute is phosphorus, the very element the process is designed to remove. This apparent paradox is easily understood when the details of the process are considered. At the beginning of the blow, SiO2 and MnO, as fast as formed from the oxidation of silicon and manganese. unite with the CaO (lime) to form a double silicate of lime and manganese (and of course iron); and since lime is in large excess, a highly basic slag, capable of holding phosphorus in solution as phosphate of lime, is formed from the start. When silicon, manganese and carbon are eliminated and the flame drops, the blowing is continued long enough to oxidize the phosphorus, which is not rapidly removed as long as carbon, silicon and manganese are present, since the affinity of those elements for oxygen at high temperatures is greater than that of phosphorus. The combustion of this phosphorus to P₂O₅ produces a great deal of heat, and the iron is protected from oxidation by the phosphorus as long as any considerable amount of the latter is present, just as in the acid process the iron is protected by silicon, manganese and carbon. Upon completion of the necessary length of "after blow" to remove phosphorus, the vessel is turned down, and the steel handled as usual.

As phosphorus is the source of much of the heat necessary to maintain the carbonless iron fluid at the end of the blow, there must be a considerable amount of it in the pig iron used. Actually from 2.5 per cent. to 3 per cent. is needed to produce sufficient heat; and as the highly phosphoric slag produced is a valuable by-product, being sold for fertilizer, the amount of phosphorus in the pig is kept as high as it is possible to have it without too much prolonging the blow.

The removal of sulphur in the basic process is only partial and rather erratic, so that the pig iron required is not very high in this impurity.

In America, there are no iron ores so far opened up that will produce pig iron suitable for carrying on the basic Bessemer practice at a profit; no ores being mined that can be used for the manufacture of low silicon—high phosphorus pig at a sufficiently low cost. Some of the Alabama ores come the nearest to it, but the phosphorus in the pig iron smelted from them is not high enough for the basic Bessemer process, nor is the silicon low enough. Indeed, these ores produce a pig difficult to work even in the basic openhearth process, as the pig contains too much silicon, which is a handicap in basic open-hearth practice, for the same reason as in basic Bessemer; namely, because it requires a great deal of lime to slag the SiO₂ produced, and the heavy blanket of slag makes it difficult to work the heats. "Duplexing" has successfully handled these pigs; the silicon, manganese and carbon being removed in an acid Bessemer converter, and the carbonless steel then purified of phosphorus and sulphur in a basic open-hearth furnace.

As the basic Bessemer process has been several times attempted in this country, and uniformly proved a financial failure, we shall not again refer to it in these pages.

CHAPTER V

THE OPEN-HEARTH PROCESS

The open-hearth or Siemens-Martin process for making steel was developed in England about 1867 by Sir William Siemens, and independently in France by Martin. The process consists essentially of the melting down of steel scrap and some pig iron in a furnace of the bath type, oxidizing out the carbon, silicon and manganese of the metal by means of additions of iron ore, and adjusting the composition by proper additions.

In order to maintain in a fluid condition the nearly pure iron produced by the elimination of the metalloids from the bath, a high temperature has to be attained in the furnace, much higher than can be secured by the direct combustion of coal. To meet this essential condition, the regenerative principle was invented. By the use of checker chambers such as have been described in the chapter on the crucible furnace, producer gas, resulting from the partial combustion of coal in a gas producer, is fed very hot to the hearth of the furnace, meets sufficient preheated air for its combustion, and heats the furnace chamber and bath to a very high temperature. So intense a heat can be attained that when a furnace is "hot," the operator has to be constantly on his guard lest the brickwork be melted away.

Besides producer gas, other fuels can be used. Natural gas, when obtainable, is excellent for the purpose and does not have to be preheated, but is piped directly into the hearth, only the air passing through regenerators. Fuel oil is used in the same way. On the continent of Europe, coke-oven gas has been tried, and it is now being successfully burned in this country and of late furnaces have been fired with powdered coal.

Whatever the fuel, and whatever the method of working, conditions in an open-hearth furnace are pretty much the same. The reasoning that has already been given in discussing basic Bessemer steel making shows us that a basic lining and a basic slag are necessary, if dephosphorizing of impure material is to be attempted. As in Bessemer work, therefore, we have to consider both acid and

basic methods; and as both are used in the steel foundry to a large extent, a somewhat extended consideration of each is necessary.

In the acid open-hearth process, pure materials have to be used in order to produce pure steel. Low phosphorus pig and scrap are melted down in the furnace, losing part of their carbon, silicon and manganese (and some iron) in melting, and forming a slag of FeO. MnO and SiO₂. There is a very slight gain in phosphorus as in the other acid processes due to loss of iron, etc., and when producer gas is used as fuel, there is also a gain of from about .or per cent. to .03 per cent. in sulphur, which is picked up from the gas. When the metal is all melted and covered with the fluid slag, iron ore is thrown into the furnace in proper amounts, and a slag very rich in FeO is the result. The excess oxygen of the flame, and the oxygen of the iron ore, oxidize carbon, silicon and manganese in the bath. the last two entering the slag as oxides, the first boiling out as CO. The bubbling produced by the evolution of CO keeps the bath in circulation, and aids in heating up the metal. As a bath must boil vigorously in order that it may readily be heated up, some pig iron to provide carbon is essential. The silicon and manganese of the pig are also an advantage, since they tend to protect the iron from excessive oxidation in melting down. The ore is added in amounts sufficient to run the carbon down to about what is desired in the finished steel (the silicon and manganese having been first removed), and the good melter does not add so much ore to his heat that the carbon "races" down, but brings it to what he desires by progressively slower steps. In this way the last 10 or 15 "points" of carbon are removed very slowly, and probably are oxidized mostly by the excess oxygen of the gases. By this procedure the available iron oxide of the ore is largely used up, and as much of the oxide as possible boiled out of the steel. If the bath is hot enough, the usual recarburizers are then added, and the heat tapped and poured.

In basic practice, limestone is charged with the pig and scrap, and lime is used freely with the ore, whereby a basic slag is maintained throughout the process, and phosphorus largely eliminated. Sulphur is also removed to some extent, but this action is erratic and not as much sulphur is removed as phosphorus. If very low phosphorus and sulphur are desired, the steel is generally run down to very low carbon, about .06 to .08 per cent., and subjected to the action of a very limey slag for some hours. In making steel of ordinary grade, the heat is tapped soon after the desired carbon is reached. In any case, however, the slag in the basic furnace con-

tains a great deal of FeO throughout the process, and hence is strongly oxidizing. The steel, therefore, is never so well cleared of oxide as in acid practice. Moreover, in order to avoid "rephosphorizing," the recarburizers are generally added in the ladle, so that they have less time to act in reducing oxides than in the best acid practice.

Open-hearth steel, therefore, especially basic steel, contains oxides to quite a large extent; but probably not as much, even in the case of basic open-hearth, as Bessemer converter metal. The quality of the steel produced, is therefore, generally intermediate between that of electric or crucible steel, and that of Bessemer metal.

The comparative freedom of open-hearth, especially acid open-hearth steel, from gases and oxides, would make it suitable for the production of small, intricate castings. But, as we have stated in the introductory chapters, the rather low initial temperature of the metal, and the size of the heats, which in ordinary practice are tapped out in one lot and have to be poured off before the metal has time to chill, forbid the use of the process for any but heavy castings. For light work, the process as generally carried out is at a prohibitive disadvantage as compared to Bessemer, crucible or electric methods. In a previous chapter, we have mentioned the attempts to use very small open-hearth furnaces for light work such as these processes handle so well, and have indicated that though the small open-hearth furnace does handle lighter work than its big brothers, yet it is not suited to such small castings as the Bessemer, crucible or electric-furnace process.

For castings averaging over 100 or 200 lb. each, the advantages of the open-hearth process far outweigh its disadvantages, and make it the most suitable process for such work. The quality of the steel is quite high enough for the castings desired, and it will pour them readily; the cost of the steel is low; the output is very large, so that the total profit on a year's output of comparatively cheap castings is considerable; and the process can be worked with a variety of fuels and raw materials.

The disadvantages of the process are the lack of flexibility in making different kinds of steel at one heat, which has already been discussed; the heavy installation expense; and the fact that the furnace must be kept going at full capacity day and night. These disadvantages tend to make the process unsuitable for the foundry producing light castings, but for the tonnage foundry they are either unimportant, or positive advantages. For such work, the lack of flexibility

is of little moment; the heavy installation expense is distributed over a great tonnage; and continuous operation is a necessity in a tonnage shop in order that the interest and depreciation cost of the plant may be spread over the largest possible output.

THE FURNACES

The scope of this work does not permit of an extended discussion of the design of furnaces now in use, but a few general remarks upon this head must suffice.

The essential parts of a regenerative furnace are the enclosed working chamber, or hearth, with a bottom lined with refractory material, where the flame is produced and heats the charge; the four regenerative chambers (sometimes only two, when only air is preheated), filled with the brick "checker work," where the heat of the products of combustion leaving the hearth is absorbed, to be returned

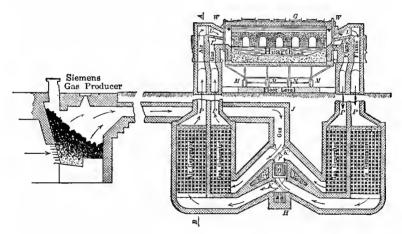


Fig. 9.—Diagram of open-hearth furnace. From Howe, "Iron, Steel and Other Alloys."

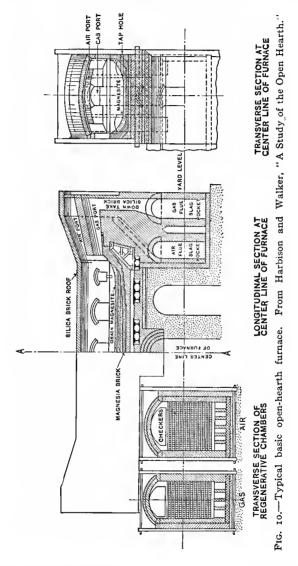
to the incoming gas and air when the direction of flow is reversed; proper passages or flues connecting the hearth with the regenerators, and the latter with the gas producers; throttling valves to control the amount of gas and air admitted to the regenerators; reversing valves to admit air and gas to each pair of regenerators alternately; and a stack of sufficient size and height to carry off the cooled gases leaving the regenerators. The general arrangement of the component parts is too well known to require extended description.

The principal changes that have been made in the building of furnaces since they were first introduced are the increase of size and capacity, the moving of the regenerators back from under the ends of the furnace, the design of special gas reversing valves, the measures that have been taken to protect door frames, ports and other parts from burning out (water cooling), and the use of tilting furnaces. Furnaces are now built up to 60 tons capacity for ordinary practice, and for the special Talbot process 200-ton furnaces are in use. The regenerators, when placed directly under the ends of the furnaces with the uptakes leading vertically from them, fill up rapidly with dust, slag, etc., and require frequent cleaning and renewal. By placing open chambers at this point, called "slag pockets," and setting the regenerators back of them, the dust and slag are collected in a convenient place where they can easily be removed, and the life of the checker works is greatly increased. During the past year, several new designs of gas and air uptakes and ports have been developed that are based upon an entirely new idea in open-hearth construction, and that promise to increase the efficiency of the open-hearth process and greatly expedite melting. This principle is being developed for various fuels, and furnaces of the new design are now undergoing careful trial. The general scheme is to mix the fuel and the air before they emerge into the hearth, instead of introducing them separately, and thereby to make the flame more like that of the blowpipe or torch. Special flues are also provided, which are closed at the end of the furnace where the flame is entering, and opened at the outgoing end, whereby the products of combustion are slowed down as they leave the furnace, with the result of improving the heating efficiency and prolonging the life of the refractories.

A few of the essential points that should be insisted upon in the design of a furnace are as follows:

Man holes should be provided for each flue, gas as well as air, between the reversing valves and the checkers; and a man hole in the stack flue, between the reversing valves and the stack or a door in the base of the stack—if the latter is far from the valves, both will be desirable. There should be ample space around the valves so that they are easily accessible. The checker chambers should be amply tied together; insufficient bracing results in a checker chamber roof so leaky that no amount of grout poured into the cracks will stop the leaks.

In the superstructure of the furnace, the roof should be entirely separate from the side walls; some builders will practically rest the roof on the walls, either by setting the "skew backs" directly on the walls, or by supporting on the walls the channel irons that hold the



skew backs. The result is that the wall cannot be cut away to patch it when worn thin without danger of bringing down the roof. The channels that hold the skew backs should be clamped to the buck

stays of the furnace so that the roof is supported entirely by the framing.

It is well to avoid "dry" door frames; water-cooled frames much more than pay for themselves in less frequent renewal of frame and doors and in the comfort of the men. The door frames should be so designed as to be readily replaced. Some builders, if allowed, will hold them in place with the buck stays. Water-cooled "ports" (or, more properly, the space between gas and air flues where they enter the hearth), if properly designed, are well worth while; if badly designed, for instance by merely running water pipes through from front to back of the furnace, they are not worth their expense. The pipes will protect but a very small thickness of brick, so that the port cuts back to them as rapidly as if not water cooled at all. Then the bricks above and below each pipe cut away, leaving a sort of shelf of hanging bricks. This shelf soon falls and exposes the pipe, which then burns away in a very few moments and is of no further service.

If possible, even in a very small furnace, there should be one or more doors on the tap hole side. Even if kept bricked up and opened only when making bottom, these doors are a great convenience, and make it possible to be sure of the condition of the banks under the charging doors. Without them, the sand or dolomite used to patch the front banks has to be rolled down with a sand spoon, and the men cannot see whether the holes are properly filled or not. Under such circumstances, many holes will not be filled as they should be and break outs will be unpleasantly frequent.

The tap hole should not be placed so low that when the bottom is made the slope to it is very steep. If placed too low, it is well-nigh impossible to scrape it clean after a heat has been tapped, and the slag and drainings of metal running into it will result in the freezing of the hole and numerous "hard taps." It should, of course, be so placed that practically everything will run out when the heat is tapped, but should be so located that when the furnace is empty the last drainings do not run to the hole, but have to be splashed out with a rabble.

The height of the furnace from hearth to roof should not be too great; the working of some furnaces is greatly improved by taking off a foot at this point. The "head," or opening where the air comes into the furnace, should be ample to take waste gases down through the air checkers as well as the gas checkers. It is essential that the hot gases should go down through both sets of checkers, and con-

tracted passages prevent this. If the air checkers are too cold it is impossible to get the furnace hot.

Too much iron or steel plating outside the front and back walls is a nuisance, as it renders it almost impossible to get at the walls to patch them when they grow thin. Bare walls as far as possible are to be preferred.

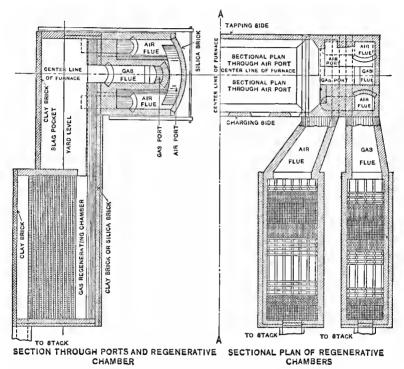


Fig. 10A.—Typical basic open-hearth furnace. From Harbison and Walker, "A Study of the Open Hearth."

Double jambs and arches for the charging doors are essential, especially double arches. When a single arch burns out and falls, it brings the wall over the door with it; a double arch, when the lower course falls, leaves the walls supported, and a new arch can be put in at the end of the week.

The beams supporting the working floor should never come close past the front of the furnace, or near the buck stays. If the main floor beam is close to the furnace, the metal running from a break out may cut into it seriously and, worse still, may form a great chunk of cold steel welding floor beam and buck stays into an inextricable tangle. There should be an open space in the floor, below the front of the furnace, large enough so that a break out may run into the pit under the hearth without striking obstructions. This hole can be kept covered with loose plates.

The throttling valves controlling the admission of gas and air generally consist of a simple "saucer" lowering into a seat, and require no description. Reversing valves, which were originally of the "butterfly" type, are now made in numerous styles, some of which are very complicated. The butterfly valve, especially if provided with a water-cooled seat, does very well for the air, but its strong tendency to warp and leak make it unsuitable for the gas. Of the many designs of gas reversing valves, some are open to the criticism that they warp and stick, while others give trouble from accumulations of soot. The rule that the simplest design is the best cannot be followed in the case of these reversing valves, since those that give the least trouble are generally rather complicated.

When producer gas is to be used, the gas mains, valve chambers of regulating valves, and gas reversing valves, should be provided with clean-out doors of ample size for the removal of soot.

Tilting Furnaces.—There are two types of tilting open-hearth furnaces, known as the Wellman and the Campbell, in both of which the hearth portion of the furnace tips up for pouring. In the former, the furnace is tipped forward upon rockers like a rocking chair, while the latter is revolved by means of rollers under the rockers, the center of the gas port opening being the center of rotation. As a result of the motion of the Wellman furnace, the gas port is brought out of line with the gas flue in the stationary end of the furnace, so that while the furnace is tipped forward gas has to be shut off, and the furnace is greatly cooled down. This cooling off wracks the brickwork severely. The Campbell furnace, on the other hand, can be left tilted over and fired as long as necessary, and the furnace is not cooled off in tilting.

Firing the furnace while it is tilted over is resorted to chiefly in continuous processes, where molten pig is poured into a bath containing a great deal of hot ore or scale. The violence of the resulting boil makes it necessary to tip the furnace in order to keep the bath from running out of the doors, and in some cases it is an advantage to be able to fire the furnace while a strong boil is going on. If the tilting furnace be used in the ordinary manner, or if the tilting

feature be taken advantage of only to pour successive small lots, it is not so essential to be able to fire while the furnace is tilted.

Tilting furnaces are more expensive to install and maintain than stationary furnaces, and should not be used except when there is good reason for incurring the extra overhead and operating expenses. The fact that occasional small lots of steel of special composition have to be made does not justify the installation of a tilting furnace, as should there be enough of such work to make it worth going after, a special small furnace can be installed to take care of it. It might in special cases be desirable to take off the heats in successive small lots, and this could best be done with a tilting furnace. The only alternative would be to use a furnace with several tap holes at different levels, which would be an awkward thing to handle, especially as the slag would have to be all taken off with the first ladle-full of steel. In general, however, tilting furnaces are not much used for steel foundry work.

For a furnace of 15 to 20 tons capacity, the crew on each shift consists of two or three furnace men, generally known as first, second (and third) helper, a gas man and helper (if producers are used), and a melter, who may have charge of five or six furnaces. When there are several furnaces, a ladle man and one or two helpers take care of the ladles; for a single furnace, the furnace men can do this work. When the furnace is charged by hand, the crew naturally has to be larger than when a charging machine is used. A 5-ton gas furnace, charged by hand, can be run with six men, who will charge the furnace in from 20 to 40 minutes. The floor crane is then, of course, used to swing the piel for charging heavy chunks. Such a crew would consist of a melter and two helpers, two gas men, and a ladle man, on each shift.

Lay-out.—The lay-out of an open-hearth shop, involving the handling of great quantities of materials, must be considered carefully in building the plant. If gas producers are used, the coal cars should if possible be able to dump their coal directly to the place where it is wanted, without any hand shoveling. Furnace space and gas producer space should be arranged so that in hot weather the sides can be thrown wide open to let in all the air possible. The open-hearth building should be high and airy, and plenty of floor space in front of the furnaces should be provided. If a charging machine is used, the wide track necessary compels the designer to make this space ample.

Accessibility of all parts of the furnace, valves, flues, etc., is an

essential feature of the general lay-out. In view of the fact that checker works have to be frequently blown out to get rid of accumulated slag and dust, and that when the furnace is being repaired many of the checker brick are taken out and replaced, it seems superfluous to state that the checkers should be entirely above ground. So arranged, and with a shop that can be thrown open to the air, checkers can be cooled off rapidly, so that the time consumed in top repairs and changing checkers is reduced to the minimum. Yet in numerous instances furnaces are built in a sunken pit like a crucible furnace, and to make matters worse, in many such cases only a very small free space is left at the front and sides of the checkers.

The floor should be commanded by a crane of ample capacity to handle any lifts needed. Generally it is better not to span both pit and charging floor with the same crane.

The ladle should never be down in a pit; if the furnace is built with checkers all above ground it will not be necessary to have it so, and the ladle will even have to be raised above the casting floor level on a car or a saddle. With this construction the space provided for the slag that slops over the ladle, and for catching slag or steel boiled out of the doors by incautious oreing, or resulting from a break out or a leaky ladle, is easily got at and cleaned out; and any skull from a break out or leaky ladle can be hooked to the pit crane and dragged out. The dragging of a heavy skull out of a steepsided ladle pit is a job that few care to undertake a second time. Until he saw a heat break out of the tap hole and go into the pit because "the ladle was not ready," the author had supposed that any furnace man would arrange his shop so that the ladle could be heated up in its proper position under the spout. The oil or gas burner for heating the ladle should be mounted in such a position as to be swung over the ladle and away again readily, and the ladle should be in position under the spout as soon as the steel is melted so that it is ready if for any reason (such as a break-out), the heat has to be tapped on short notice. For the same reason, the stopper rod should be placed before the ladle is heated up. In order to dry out the nozzle properly, the stopper rod should be raised to its highest position and clamped there. With this arrangement, it is but the work of a moment to drop the rod and have the ladle ready.

Yard arrangements for an open-hearth shop are generally quite easy to make, if ample space is available. The straight-line plan is of course the most desirable. Either a locomotive crane or overhead cranes may serve the yard, preferably the latter. Magnets should be used for unloading pig and scrap, and for loading furnace charges on the narrow gauge cars sent to the furnaces. If the furnaces are machine charged all material will be loaded in charging boxes and the open-sided "piels" used for charging ingots and heavy crop ends. The magnets can load these very handily. For hand-charged furnaces small flat cars serve the same purpose, and are readily loaded with magnet and crane. Generally, unless a favorable difference of level can be taken advantage of, the furnace charging floor is considerably above the yard. Elevators or the cranes can be used to lift cars to the furnace floors if only one or two furnaces of small capacity are installed, but for any considerable tonnage they will prove rather slow, and it will be better to run the cars up an incline to the floors.

Starting a Furnace.—Operations of course begin with a new furnace, and the first thing is to get the furnace heated up and set a bottom. It is especially important to heat up a new furnace carefully and slowly, to guard against the spalling and cracking of the silica brick of which it is built. First throw the reversing valves on center and close the air and gas inlet valves; then build a wood fire in the hearth and feed it constantly (or use a hard coal fire in a stove). Build a second fire at the base of the stack to create the necessary draught. Have a pile of sand ready for use, and if the fire gets too lively and the flames touch the roof (before the roof has become hot), shovel in sand to deaden the fire. Keep the fire going night and day till the furnace walls and roof are thoroughly dry and hot.

A better and quicker way to warm up a furnace is to use oil burners introduced through the doors, or through the gas ports. If natural gas or oil be the fuel, the main burners of the furnace can be used for warming up; they can best be lighted by means of a small wood fire in the hearth. In heating up with oil or gas burners, care must be taken to bring the furnace up very slowly, turning on the fuel sparingly at first.

In one or two days the furnace will be hot enough to light producer gas with a wood fire; oil burners will reduce this, or get the furnace up to much greater heat in the same time. Have the producers making good gas before any attempt is made to turn gas into the furnace. Throw the reversing valves over, admit gas cautiously to the furnace and keep the wood fire going until the gas lights. When a furnace is not very hot it is well to open the doors when gas is first put on. At first there will be only a fitful dull red flame

almost lost in the black gas. The first reversals should be made carefully, at intervals of about $r\frac{1}{2}$ hours, preferably shutting off the flow of gas before reversing in order to avoid hard kicks. On the first few reversals, new fires may have to be made to relight the gas. After a half dozen reversals the gas burns freely, the furnace begins to pick up heat rapidly, and should be "thrown over" at shorter intervals. After about 24 hours air can be admitted and the reversals made every 20 minutes or half hour.

As the furnace heats up, the roof begins to expand. The longitudinal expansion is largely taken care of by the "slip joints" provided for the purpose, but the horizontal motion must be taken up by loosening the nuts on the tie rods a little at a time as the roof moves. The men go over every nut at intervals, and loosen them up enough so that they move easily, but not enough to loosen the roof and allow bricks to fall out. Here and there a few will slip out of the roof, and these must be replaced with others, held in place with wedges until the expansion is sufficient to hold them securely. When the furnace has reached full heat, the making of bottom begins.

The Acid Bottom.—In the acid furnace, the roof, walls and bottom are of silica brick and a lining or bottom of silica sand is set on the bottom brick. It is a saving of sand to bank a good layer of silica brickbats, left from the construction of the furnace, in the angles between sides and bottom. Over these a layer of sand about 2 in. thick is shoveled, and the furnace run up hot enough to partially fuse or sinter this sand into a solid mass. As soon as this layer is firm, another of the same thickness is added, and so on until the bottom is brought up to its full dimensions.

The tap hole can be broken through with bars from the outside after the bottom is made, or an iron pipe the size of the hole desired can be used as a form. In this case the tap hole is shaped with siliceous ganister and closed with coal and sand before bottom making begins.

The sand used for bottom making (and for patching the bottom after each heat) should be either a silica sand with just enough FeO and Al_2O_3 to make it "set" (or partially fuse), at the full heat of the furnace, without getting too soft, or preferably a pure silica sand mixed with an impure one containing a larger proportion of FeO and Al_2O_3 . It is generally much easier to obtain a pure sand and an impure one to mix with it than to get just such a sand as is needed.

When the bottom is finished, a "wash heat" of acid slag may be

melted on it; or a light heat of pig iron and scrap melted down, and the slag made bulky with iron ore and sand. In any case, the slag should be thoroughly splashed up on the banks in order to soak the bottom full of it. The furnace is then ready for melting steel.

The tap hole is closed with fine anthracite coal mixed damp with a little sand. The men on the charging side run a rabble or splash hook into the furnace and block the inside of the hole, while two men do the closing from outside. Coal and sand are thrown up into the hole and packed firmly in place with a rammer until it is full. Sand is then banked over the coal on the outside and a few shovels of "seal" (coal ashes and sand in about equal parts) are thrown over the hole on the inside. Great care should always be taken to scrape the hole clean of slag, and especially of metal, before closing, as the penalty for carelessness is a "hard tap."

Needless to say, the tap hole of a tilting furnace is easily closed and not subject to hard taps. Unless it is intended to roll the furnace very far over to keep metal and slag from boiling out the doors, the hole is very lightly plugged; generally burlap bags are used, which stay in place as the furnace is tipped up until the slag is above the top hole, so that steel, not slag, is the first to run out of the hole.

The Basic Bottom.—The bottom bricks of the basic furnace are of magnesite, since acid (silica) bricks would react with the basic bottom material and produce slag. Of late years what are known as "Metal-kase" bricks are often substituted for regular magnesite bricks. These are open-ended square tubes of light sheet metal, filled with magnesite tamped tightly in place. In certain cases, it is more economical to use them, especially for bottoms. two courses next to the steel shell may be of clay brick. The walls and roof are of silica brick, and to prevent their slagging the magnesite of the bottom bricks at their contact, an action which would be increased by the iron oxide and manganese oxide of the furnace slag which spatters upon the walls, one or two courses of chromite brick are sometimes used to separate the bottom and the wall bricks. Formerly, this was always done, but of late years many furnaces have been built with the magnesite and silica bricks in contact. Chromite brick are sometimes also used to pave the bottom of the gas port. These bricks are made of chromic iron ore, a double oxide of iron and chromium which is almost absolutely neutral, and hence are not slagged by either the silica wall brick or the magnesite bottom.

The bottom of the basic furnace may be made of either calcined dolomite (CaO MgO), or calcined magnesite (MgO), or both. Magnesite is greatly to be preferred, as it makes a much denser bottom that can be confidently relief upon to stay in place when once set in and soaked full of slag, while dolomite bottoms frequently give trouble by coming up in patches, or cutting through. Whichever is used, it should be mixed with sufficient basic open-hearth slag (generally from 5 to 15 per cent.) to make it sinter at the full heat of the furnace, and placed a layer at a time as an acid bottom is set. Sometimes a little tar is mixed with the magnesite and slag. Magnesite brickbats can be used to help out in filling up the corners, before any lining is placed. During the war we all learned to our sorrow that the best magnesite comes from Austria, and that it is almost impossible to so modify the composition of our domestic mineral as to make it wholly satisfactory. The Austrian product contains just enough FeO and SiO2 as impurities to make it sinter at a proper temperature; in an effort to improve the qualities of our domestic magnesites, many of which are almost pure MgO, iron ore was thoroughly mixed with it before calcining. In spite of all efforts, however, this prepared material was less satisfactory than the foreign product; and some of the Canadian magnesites which contained 15 to 20 per cent. lime were even worse. The analyses of several lots of magnesite are given in the chapter on the electric furnace.

Calcined dolomite slakes like burned lime upon exposure to the air. For this reason, dolomite in open-hearth practice is sometimes used in the raw state; when the calcined material is used, only a small supply should be kept on hand, as it deteriorates badly in storage.

The patching of the slag line is done with dolomite or magnesite, frequently both, the former predominating. Holes in the bottom that have to be splashed out and patched are best repaired with magnesite. The tap hole is closed with either one, mixed with coal, but magnesite is preferable, as a dolomite hole is very uncertain, frequently taps itself, and nearly always makes the men jump lively in tapping.

Week-end and General Repairs.—On Saturdays, the furnace is gone over and minor repairs and patching done for the week. Generally the ends of the furnace are opened, the accumulation of slag chipped out of the gas flues back of the ports, and the slope of the gas port filled up with sand (in acid furnaces) or magnesite (in basic). New doors may be required, or patches in the walls, door jambs and

arches, etc., when the brickwork begins to get old. Accumulations of dust in the flues between reversing valves and checkers, or in the stack flue, may have to be removed; and in general things are made ship-shape for the next week. When producer gas is used, the gas mains and valves are burned and scraped free of soot at the same time; with the old fashioned gas producers, too, the accumulated ashes were removed on Saturdays. A modern plant, of course, would have producers that can be cleaned at any time.

When a producer gas furnace of the usual type has been used for some time, and the brickwork has grown thin, the gas often begins to leak through the roof arch of the gas port, generally where it joins the gas uptake, and to mix with the air in the "head." This condition can usually be detected by the flame playing out where the roof and the end wall of the furnace join, and must be attended to immediately or the brickwork will be cut away rapidly and the furnace put out of business. The end wall is partly taken down to get at the trouble, and the holes in the brickwork that are making the trouble are plastered up. The best material for this purpose is chrome ore ground fairly fine and mixed with water and a little fire clay to form a stiff mud. The leaks must be thoroughly stopped, but of course when a furnace gets to this point it is a question of only a short time until a new top is needed.

When the walls have worn too thin to admit of further patching and the ports have cut back until the furnace no longer heats properly, a short shut-down is necessary. If the roof is fit to outlast another set of walls and ports, it can be left standing, or sometimes part can be repaired. As soon as the furnace is cold enough to work on, often before it has grown black, the walls are knocked out and removed, the brickwork of the ports and ends is dug out as far as necessary, and the bricklayers begin rebuilding as soon as it is humanly possible to do so. At the same time the slag pockets are cleaned out, and the checkers examined. The latter may have to be removed altogether, or part of them taken out, down to good bricks. Checker bricks that are thoroughly glazed are very inefficient and should not be put back in place. A shut-down for this partial repairing keeps the furnace idle altogether about a week. Fuel is put on again as soon as possible and operations resumed.

If the roof has fallen in while a heat was molten, as sometimes happens when the furnace is run close to the limit of the top, it is not necessary to get the metal out. The new walls and roof can be put up as if the furnace were empty, and the skull melted out when

gas is turned on again. It will, of course, take the furnace a bit longer to become cool enough to approach, under these circumstances.

Break Outs.—A break out is a mean thing to contend with, yet if possible it should be fought. An approaching break out can often be foreseen by the violent boiling of the bath over the spot where the bottom is being cut through. Frequently, if at the front of the furnace, the first warning is the reddening of the hearth plates at the threatened spot.

As the bath heats up and grows comparatively quiet, one spot should maintain a steady slight boil—the spot right over the tap hole. If this boiling ceases, a hard tap is to be expected. If it becomes violent, the bath is cutting its way out the tap hole and will soon be running down the spout, "self-tapped." A break out when the heat is about ready, is best fought by tapping the heat and taking chances on analysis, adding silicon and manganese largely on rapid calculations, frequently made in the head of the melter as he hastens to the scene of action. An approaching break out in the flat part of the bottom cannot be fought, and the only remedy is to tap at once before the steel goes into the pit and renders it impossible to approach the tap hole. Small break outs through the banks, at or near the surface of the bath, are fought by shoveling sand and bricks (in the basic furnace, dolomite, bricks, etc.) into the opening with the sand spoon, and turning the hose on the plates on the outside to chill the metal. With all hands from several furnaces on the job, a very mean looking break out can be stopped, but it is hard work and men play out left and right.

A break out, of course, is caused by an improperly closed tap hole or a badly repaired bottom. To guard against break outs in the bottom, large holes should be splashed clear of metal after the heat is tapped, and filled up with sand (in the basic furnace, magnesite), well set in, a layer at a time. Break outs through the front banks are more easily guarded against when the furnace has doors in the tap hole side, through which the front is repaired properly and its condition examined. It is impossible to overemphasize the importance of splashing all the metal out of holes in the bottom before any sand or magnesite is put in to fill them up. If metal is left in a hole, the thin layer of bottom over it will come up on the next heat, and the hole will grow larger; it is only a question of time and not such a long time at that, before a hole in which metal is left after each heat will grow very large, and sooner or later a heat will go

through the bottom. Those who have seen this happen are not likely to wish to see it a second time.

Tapping.—In order to get the heat out in a lively stream from the first, the tapping of an open-hearth furnace should be done carefully. Two men are required at the hole, one armed with a narrow shovel and one with a tapping bar. The banked-up sand should be removed cleanly and the filling of the hole dug out over the whole cross-section, until a large red-hot patch is exposed. vigorous punch of the bar into the center of the filling will then bring the steel quickly, and the stream will almost at once cut out the whole hole. In some cases the bar may have to be driven through with a sledge. At times the steel comes almost of itself, and the men have to work fast and be nimble in getting out of the way as the stream breaks out. The men tapping should have a clear floor left and right to give them space to get away at once in case of too lively a tap. Of late several ingenious tap hole guards have been devised and described in the technical press, for the protection of the men tapping a furnace from the showers of sparks when a heat "taps itself," and their use is strongly to be recommended.

If the tap hole does not at once cut itself out, the melter or man in charge gives the word to "punch her out," whereupon the men on the charging side, who have a heavy bar ready for the purpose, slide this bar in through the peep hole of the door and punch out the hole with a few vigorous strokes.

A hole that cannot be punched through by hand must be opened with the heavy tapping bar, sometimes called the "snot bar." This tool is made of about $1\frac{1}{2}$ -in. round steel, and is provided with a handle which can be slipped over it and secured with a wedge. It is driven through the tap hole with sledges, the handle attached and wedged fast, and is then backed out of the hole by sledging on the handle. When the steel comes, quick work is necessary to save the bar.

If this procedure will not open the tap hole, it is because steel has run into the filling and frozen, and it is necessary to burn it through. This was formerly done with an oil burner, the steel being heated until it started to melt, after which the oil was shut off and a strong blast of air alone directed upon the melting metal. The air oxidized enough of the iron to FeO to keep the steel at melting point by the heat of combustion of the metal itself, and thus a hole was melted through the plug and back to the bath.

This was a slow method, and the heat could not always be saved,

even by additions of ferrosilicon, pig, etc., to keep it at the proper composition. The banks of the furnace were badly cut into, and sometimes the proceedings were enlivened by a break out. To-day a hole plugged with steel is opened quite quickly by means of a bottle of oxygen, a length of rubber hose and a piece of $\frac{3}{16}$ -in. iron pipe. The pipe is put into the furnace until it is hot at the end. It is then withdrawn, and a gentle stream of oxygen run through it, whereupon the end of the pipe begins to burn. By pressing the end of the pipe against the steel in the hole and turning on more oxygen, the metal is burned and melted through as is done with the oil burner and air blast. The whole procedure, however, takes only a few minutes when oxygen is available.

Burning through a hole is a lively job, as the operator is exposed to a shower of slag and scintillating steel that pour from the hole and keep him dodging, and without his glasses he is in danger of having an eye badly burned. Such a badly plugged hole means that it was not scraped clean and carefully closed after the preceding heat was tapped.

The Acid Process. Melting.—For the usual pig and scrap method in the acid furnace, the charge consists of from 20 to 40 per cent. pig and the rest scrap. Much less pig than this can be charged, but in order to get a bath composed largely of scrap, which necessarily melts very low in carbon, hot enough to tap, a good deal of pig iron has to be thrown in to make the steel boil.

As this is a discussion of the steel foundry, the all pig-iron process (either acid or basic) is automatically excluded from consideration, since we shall have to remelt in every heat from 20 to 40 per cent. of our own scrap, the proportion largely depending upon the size of our castings and the number and size of the heads used on them.

The high-silicon content of the foundry scrap which we have to remelt greatly affects the calculation of the charge. It is in general the best practice so to proportion the silicon and manganese of the charge that these elements will be practically eliminated when the bath is melted. In a producer gas-fired furnace, we can assume a good composition for the charge as carbon 1.20 per cent., manganese 1.00 per cent., silicon .50 per cent. In an oil furnace, the silicon, manganese, and sometimes carbon, must be higher. Assuming that we have 30,000-lb. charges, one-third our own scrap (carbon .25 per cent., silicon .30 per cent., manganese .60 per cent.), pig iron at carbon 4 per cent., silicon 1.5 per cent., manganese 2.0

per cent., and low phosphorus scrap at carbon .15 per cent., silicon .10 per cent., manganese .40 per cent., let us see the proportions of pig we shall have to use.

| Material | Weight | Weight, lb. | | | |
|-----------------|-----------------|-------------|-------|------|--|
| | | С | Si | Mn | |
| Our own scrap | 10,000 | 25 | 30 | 60 | |
| Purchased scrap | 12,000 | 18 | 12 | 48 | |
| Pig iron | 8,000 30,000 | 320 | 120 | 160 | |
| | 30,000 | 363 | 162 | 268 | |
| | | 1.21% | - 54% | .89% | |

Such a mixture will melt quite free from silicon and manganese, and probably at from .70 to 1.00 per cent. carbon. The consumption of ore should not exceed 6 per cent. of the weight of the charge, and on such heats will be found to average well below this figure. It is possible for quicker running to reduce the pig iron so that the carbon shall be about 1.00 per cent. as in the next example.

| Material | Weight | Weight, lb. | | | |
|-----------------|--------|-------------|--------|--------|--|
| | | С | Si | Mn | |
| Our own scrap | 10,000 | 25 | 30 | 60 | |
| Purchased scrap | 13,500 | 20.25 | 13.5 | 54 | |
| Pig | 6,500 | 260.00 | 97 · 5 | 130 | |
| | 30,000 | 305.25 | 141.0 | 244 | |
| | | 1.01% | . 47% | . 81 % | |

Pig iron very high in silicon and low in manganese may frequently cause embarrassment, as may be seen from the following calculation, based on pig of 3.0 per cent. silicon, .30 per cent. manganese.

| Material | Weight | Weight, lb. | | | |
|-----------------|--------|-------------|-------|-------|--|
| | | С | Si | Mn | |
| Our own scrap | 10,000 | 25 | 30 | 60 | |
| Purchased scrap | 13,500 | 20.25 | 13.5 | 54 | |
| Pig | 6,500 | 260.00 | 195.0 | 19.5 | |
| | 30,000 | 305.25 | 238.5 | 133.5 | |
| | | 1.01% | .79% | .44% | |

This condition may easily become complicated by the availability of purchased scrap consisting largely of cast steel averaging .30 per cent. silicon. The use of high proportions of this scrap, or of high silicon pig, will result in heats that melt high in silicon, on which much time will be spent in the elimination of the silicon by means of ore. Confronted with such conditions we must cut down the proportion of pig iron so that the charge will be low in carbon, yet contain enough silicon, when melted, to protect the carbon from oxidation. Our bath will then melt at about the carbon content of the charge, and we can run it down with little or no ore; or if it melts too low in carbon and is hard to get hot because it will not boil, we can "pig" it to assist operations.

Many of the low phosphorus pigs that are available in the East to-day are very low in manganese also, averaging about .30 per cent. When these pigs are used, the proportion of manganese in the bath will be low, with the result that the amount of manganese oxide available for the slag will be small. Consequently the proportion of iron oxide in the slag will be high, and the loss of iron correspondingly increased. (Campbell¹ has shown that the figure FeO + MnO in acid open-hearth slags is approximately a constant.) This may be offset by the use of manganese steel scrap, spiegel or even ferromanganese as a part of the charge. The experience of the largest producers of open-hearth steel in this country has been that the presence of from .70 to 1 per cent. manganese in the charge is highly desirable, not only to reduce the percentage of iron oxide in the slag. but also to ensure the production of first-class steel. Probably because the higher the MnO in the slag, the lower the FeO can be, and hence the more thoroughly the metal can be deoxidized, the

¹ "The Manufacture and Properties of Steel," 2d Ed., p. 272.

steel is of much better quality when the manganese in the charge is around 1 per cent. than when it is low.1

Charging ore with the pig and scrap is a possible method of hastening the elimination of silicon and manganese from the bath while melting, but is not often resorted to because of the effect on the bottom. The FeO of the ore takes up SiO₂ whenever it can get at the hearth, cuts the bottom severely and forms great volumes of slag.

In charging the acid furnace all the pig iron is placed upon the bottom, and the scrap on top of the pig. This method is followed to prevent the severe cutting of the bottom caused by the iron oxide formed on the melting scrap coming immediately in contact with the silica bottom and absorbing SiO_2 , with consequent formation of slag and loss of iron. By interposing a layer of pig, which melts first, a pool of molten metal is formed into which the melting scrap drops, and the slag floats on this bath from the first. By floating on molten pig, much of the FeO is reduced by the carbon, silicon and manganese of the pig, and the iron saved.

In some shops the scrap is charged first, with the idea that if the pig is on top it will melt first and flow over the scrap, whereby the iron of the latter will be protected from excessive oxidation by the carbon, silicon and manganese of the pig. This protection, however, is only partial at the best, and probably the advantage gained is more than off-set by the increased loss of iron oxide formed on the scrap, which comes at once in contact with the bottom and unites with the SiO₂ of the sand to form slag, before the pig iron can melt and run down to head it off from the bottom. A part of the pig, however, may with advantage be charged on top of the scrap.

Charging should if possible be completed at one operation. Sometimes when a great deal of light scrap has to be melted, it is impossible to get it all in at once, though much can be done by charging scrap into say the right-hand end of the furnace, with the flame entering from the left, then reversing the valves and filling up the left-hand end, reversing again and charging more scrap at the right. During the charging of the left side, the metal on the right is heated up and sinks down, making room for more. Too much light scrap is a great nuisance, as it not only increases the loss of metal but also delays charging and melting. Turnings are especially hard to melt, as they stick to the bottom in lumps and have to be poked off with a bar.

¹See Howe and Barba, "Acid Open-hearth Process for Manufacture of Gun Steels," read before Feb., 1922, meeting Am. Inst. Min. & Metallurgical Engineers.

In handling the open-hearth furnace, heated with producer gas, it is desirable to use an even flow of gas, and as little of it as possible. When the charge is cold, the flame will be dark at the best, and a good supply of air will be needed to burn the gas. The gas valve should be opened to about the point at which it will be kept; the stack damper raised just enough to carry away the waste products, but not enough to draw the gases so strongly that the flame will "race" through the furnace; and air enough admitted to make the flame look as white as possible. If the gas is being fed so fast that a steady flame pours out of the peep holes in the doors, especially if raising the stack damper will not keep the flame in the furnace, too much gas is being fed and the furnace being "crowded." By watching the top of tho stack, waste of gas can be guarded against. The furnace operater and the gas man have to get used to each other; if the former keeps his supply of gas regular by using a constant opening of the valve, his gas man soon learns how often he has to dump coal in the producers to maintain even pressure, and things will go smoothly. A furnace man who constantly makes large shifts in the position of his gas valve, alternately draining the producers of gas and nearly blowing the tops off them by shutting down his valve, does not know his business and his gas man will hate him.

The flame should be regulated chiefly by shifting the stack damper and air-supply valve. As soon as the charge begins to get hot (and looks red through the blue glasses), the furnace man should begin to shut off the air supply a bit at a time, and try to make the flame roll through the furnace, hanging well down into the stock, and moving slowly. The flame should look as white as possible, free from dark spots and flickering; and neither a short flame reaching only part way into the furnace, nor a flame so long as to be still burning at the far end, should be carried. The former means either too little gas or too much air, and results in heating the charge mostly at one end and oxidizing it severely; the latter means too much gas, which will continue to burn quite uselessly in the checkers and waste fuel scandalously. Above all, the flame should roll slowly along, hugging the stock and playing through it, should be white, and long enough to heat the whole length of the hearth.

With such a flame, the melting stock will "honeycomb," and melt down rapidly and evenly; it will be clear at a glance that the metal is hot throughout, a thick section being nearly as hot in the middle as at the surface. A swift, short flame, dragged along by the draught of a wide open stack damper and fed with too much air, "glazes" the stock so that it melts chiefly in a thin layer at the surface, and largely at the top of the pile. Such melting oxidizes the metal badly, wasting iron and making the steel "wild" from excess of dissolved oxide.

At all times, the furnace man watches his roof, walls, and outgoing port closely, keeps the flame down to the bath (and so away from the roof), and slacks the temperature when he sees his brickwork giving signs of starting to melt. By shutting off the air, and lowering the stack damper, the temperature is kept down.

Natural gas or oil firing should be handled along the same lines, though changes in the fuel supply are not so undesirable.

When the charge is "down," and nearly all the floating pieces have melted, the helper gets busy with his bar and pokes all scrap free from the bottom. It may take him some time to do this, especially if much light scrap and turnings have been charged, as these stick to the bottom like limpets.

When all is melted, the bath should be well stirred with the bar, and the "raise" noted. By this is meant the height to which the slag boils over the bar as it is moved about. The colder and "wilder" the metal, the higher this boil will be. The boiling produced by this rabbling has the effect of freeing the steel from oxides by bringing it into contact with the slag.

A test of the metal is then taken by means of the small test spoon, which should be first warmed in the furnace a moment, then well coated with slag, then dipped deep in the bath and rapidly withdrawn. A small test mould is filled, and the metal left over poured slowly from the spoon to judge its temperature. If it pours clean from the spoon to the last drop, it is very hot. The more of it freezes in the spoon, the colder and wilder the metal. A little of the slag adhering to the handle of the spoon is saved from each test; by its color the progress of the elimination of iron oxide is watched. When the steel is cold and wild, it scintillates and "rises" strongly in the test mould. Another means of judging the temperature of the bath is the manner in which it melts away the bar used for stirring it up; a hot heat cuts a bar off quite square, a cold one leaves the end "rat-tailed." The heat should be hot before oreing is started.

By the fracture of the test piece or by carbon analysis, the melter estimates the carbon of the bath and the amount of ore needed to eliminate the carbon to the desired extent. The helpers throw in ore while the first helper swings doors and watches the behavior of the bath, which, unless still high in silicon, begins to boil violently

as the ore is thrown in. If much ore is needed, it cannot be added all at once, or the bath will "boil over." The boiling increases in violence only slowly up to a certain point, when the addition of one or two more chunks of ore will bring the whole bath boiling up, and slag and metal will run out of the door and all over the floor. At the same time, the great amount of CO suddenly evolved (and burning to CO2) so fills the furnace with gas that the flame pours from all openings in volumes and the neighborhood becomes decidedly unpleasant. Should a "boil over" occur, the reversing valves should be thrown "on center" at once, to shut gas from the furnace, and if the boil does not subside, the gas should be shut off, the reversing valves thrown over again, and no more gas admitted till things quiet down. It does not do to leave the reversing valves on center long, as it generally heats them badly, but a "boil over" comes so quickly that to throw the valves on center is the only way to shut off the gas in time to do any good.

The boiling produced by the ore should be maintained by further additions, until in the opinion of the melter enough has been added to eliminate the carbon to the desired degree. When the first effect of the ore begins to wear off, the bath should be stirred with the bar and a second test taken. If enough ore has been charged. this second test should show greatly reduced carbon content; and a third taken half an hour after the second should show still lower. Should the second test show too little reduction of carbon, more ore must be added; the same is true after two or more tests if the carbon ceases to "slide down" fast enough, and frequently more ore will have to be added to keep up a proper rate of elimination. For the production of really fine steel, however, the effect of the ore should be pretty well worn off an hour or two before tapping, and ideal practice is to get it all into the furnace as soon as possible after melting, and have the carbon "slide down" rapidly at first, then less rapidly and then very slowly, taking an hour to make the last 8 or 10 "points." By this practice the FeO in the slag available for oxidizing silicon, manganese and carbon is practically all exhausted before the heat is tapped, and the FeO is largely eliminated from the steel by reaction with the carbon of the bath, and taken into the slag. As the effect of the ore wears off the "raise" becomes less, the slag grows less black and watery, and the metal pours more cleanly from the test spoon.

Should too much ore have been added, so that the carbon is "racing" down, the bath will be practically carbonless before it has had time to get hot, may even freeze up, if the melting has been done

with a very short, oxidizing flame. To prevent this, pig iron or ferrosilicon must be thrown in until the carbon content is held stationary. Ferrosilicon is used when the bath is hot, but losing carbon too rapidly. The silicon uses up the oxygen of the ore without making the bath boil, and generally will prevent the carbon from dropping further. Pig iron is used when the bath has to be much heated up without further elimination of carbon. The boiling produced by the oxidation of the carbon of the pig greatly assists in getting the bath hot; and, of course, if enough pig is added the carbon of the bath will be kept up.

Should a heat freeze, which happens only if such a "sharp" flame has been used in melting as to oxidize most of the carbon, silicon and manganese before the furnace is hot enough to maintain the resulting low carbon metal fluid, pig iron and spiegel or ferromanganese, are thrown in and melted, and more added as necessary until the fluid bath of metal high in carbon, silicon and manganese has taken up the solid or semi-solid metal beneath. This is a long, tiresome job, eats into the bottom of the furnace tremendously, and results in a heat poured into scrap, as the cutting of the bottom compels the tapping of the metal as soon as it is liquid enough to run out.

The temperature of the steel is followed by the indications already described. When the carbon is low enough to tap the heat, the metal should be so hot that it will pour out clean or nearly clean from the test spoon. If it will not, the temperature should be increased, holding the carbon up if necessary by throwing in pig iron or ferrosilicon, until the steel will pour reasonably clean. If the ferrosilicon, pig, or ferromanganese used as recarburizers are added in the furnace, a last test after they are melted does not come amiss. This last test, if the silicon (or both silicon and manganese) have been added, should pour perfectly clean from the spoon, and the test should lie flat in the mould.

Some furnace men add ore till the carbon is racing down and the steel full of oxides, the tests skulling the spoon and scintillating and rising strongly in the mould, throw in ferrosilicon and pig iron to check the elimination of carbon, tap the heat as soon as these are melted, add their ferromanganese in the ladle and pour their steel. They melt their metal hot and keep it hot, so that it at least pours. But it is full of oxides, "raw" and "wild" in the moulds, and of poor quality. By heavy doses of manganese and silicon, and frequently of aluminum, the steel is quieted (freed of oxide and gases) enough to make fairly sound castings; but it is not good steel. These

are the shops where the makers of "final deoxidizers" make a great hit, and they do it partly by the use of "final deoxidizers," but largely by so improving the practice that when the heat is tapped, the oxides are largely eliminated to start with.

The French metallurgists have worked out a method of deoxidizing the slag (and hence the steel) in acid open-hearth work, to an extent much greater than is possible by simply allowing ample time for the effect of the ore to wear off. This consists in sprinkling the slag, after the bath is sufficiently hot and the available oxide pretty well boiled out, with crushed bauxite or loam (containing alumina in considerable quantity), and crushed coke. The adding of these materials has to be carried out with great judgment and skill, the loam, and especially the coke, being spread evenly over the surface of the slag so that none falls in a mass at one point. This involves much dexterity with the shovel, as, of course, if coke falls in a mass at one point it is very liable to sink through to the bath and add carbon to it.

The action of these additions is to eliminate the oxides of iron and manganese from the slag by the reducing action of the coke, according to the usual reactions,

$$FeO + C$$
 equals $Fe + CO$
 $MnO + C$ equals $Mn + CO$

while the alumina replaces them in the complex silicate which is the slag. Carefully carried out, this procedure results in a slag nearly white, quite free from the brown color of FeO and nearly free from the green of MnO; though of course these oxides are not entirely eliminated. Yet skillful melters will make these additions until the slag (and hence the steel) is remarkably free from oxides; and the steel so made is of exceedingly high quality. Steel men in this country have, so far as the author is aware, never attempted (or at least never adopted) this procedure and many of them appear never to have heard of it. In these days of final deoxidizers, perhaps we shall see it used.

When the heat is tapped the furnace men scrape and close the tap hole, warm up the furnace a bit, and shovel sand all around the slag line of the hearth to repair the damage done by the slag. What can be reached by shoveling is so repaired, the out of the way spots in the corners are got at with the sand spoon. The furnace is made hot enough to "set" the sand and is then ready for the next heat.

Typical Heats.—In order to illustrate the procedure in handling

an acid open-hearth furnace, the history of a few typical heats is shown in the tables that follow. In these, the charges weigh 30,000 lb., and a "Base" of 29,000 lb., is taken as the approximate weight of steel to be obtained. The analyses given are for the charge, the steel desired, and that obtained. The symbol .04 signifies that .04 per cent. is the highest percentage of an element that will be accepted. The phosphorus and sulphur content obtained are in some cases lower than that given for the charge. This is explained by the fact that the latter is figured on known maximum contents of the pig iron and scrap used. The pig iron in this plant was kept in piles according to its phosphorus and sulphur, and each pile was known to be below such and such a figure. Similarly the different grades of scrap were known not to exceed certain figures.

The symbol R equals 1", or R equals 2", etc., signifies the height to which the bath boils when stirred, as already described. This shop was run for the production of steel upon which very close control of analysis was demanded, and a small laboratory for running "color carbons" on the tests was located in the open-hearth building. A color carbon analysis could be run in about 20 to 25 minutes, so that the men in charge were kept well posted on the analysis of the bath. To day, of course, the carbon is determined by combustion, and an analysis is run in 10 to 15 minutes.

The symbol BR represents the last test taken before the recarburizer is added. The carbon of this test has to be estimated; "carbon estimated," in the tables, refers to it. Comparison with the analytical results, in the "history of heat," shows the very accurate judgment of the mejters. These heats were made 17 years ago, in furnaces of rather old-fashioned design, burning producer gas generated in producers of types now obsolete. The time consumed per heat is accordingly much greater than is typical of modern practice, especially in furnaces burning oil. They illustrate, however, the procedure in making steel of the highest quality, and except for the slow melting and finishing can well be taken as a guide to good practice.

Heat (1) was chosen to illustrate a case in which not only was no ore added to decarburize the bath, but also, in order to get the steel hot without running the carbon down too far, a considerable amount of pig had to be added. The temporary rise in the carbon content after the addition of the pig, followed by renewed elimination of carbon, is well shown.

The slag of this heat was brown throughout. This refers to the

color of the fracture of the cold slag, and is used as an indication of the degree of oxidation of the bath. When the bath is first melted the slag is nearly black, very thin and fluid, and consists largely of silicate of iron. The steel in contact with such a slag will be full of oxide. During the "working" of the heat, the black changes to a brown and this gradually to a greenish-brown or greenish-gray color. The green shades indicate the presence of MnO in the slag, and a much decreased oxidation of the bath.

The limestone added to this heat was for the purpose of thinning the slag. Frequently, especially on nickel-steel heats, the slag absorbs so much silica from the bottom of the furnace as to be almost infusible, and in these cases limestone is used to lower its melting point, and hence thin it. When, on the other hand, the slag is too thin and watery because it contains an excess of FeO, loam is added to thicken it.

Heat (2), made in the same furnace as (1), though quiet the first time it was rabbled (that is, there was no "raise" over the bar), became very wild, with a 3-in. raise, after the addition of the ore, and the slag was very thick and had a strong tendency to remain brown. These two conditions, pointing to a bath containing a great deal of oxide, grew less marked when the steel was ready to tap. The normal loss of manganese (and carbon and silicon) was to be expected, and was obtained. Thus .75 per cent. of manganese was added and dropped to .63 per cent., and the metal was raised to .32 per cent. carbon, which dropped to .27 per cent. Reference to heat (1), in which no ore was used, shows that there was a slight gain in manganese in that heat in spite of the 2-in. raise and the brown slag, which tend to show an oxidized heat.

Heat (3) went into the same casting as heat (2). The ore was evidently added before it was known that this heat was to wait for heat (2), and had the usual effect of sending the carbon down rapidly. In fact, too much ore was added for really good practice, as the carbon was already low and an accurate estimation of it would have dictated a much smaller addition, or none at all. Had it been known at the first that this heat was to wait for heat (2), it would have been run like heat (1), no ore at all added, and the heat allowed to run down slowly. As it was, a heroic addition of pig was needed to keep the carbon up, and it was considered advisable to use 12 per cent. ferrosilicon and charge it in the furnace, in order to counteract the oxidation resulting from the elimination of the carbon to such a low point. That the silicon had the effect of quieting the

bath thoroughly is evidenced by the fact that no manganese was lost in recarburizing.

Heat (4) is in every respect a normal one. All the recarburizer was added in the furnace so that a heavy addition of manganese (1.01 per cent.) was needed, in order to obtain .75 per cent.

Heat (5) illustrates the practice of charging a certain amount of scrap (billets in this case), near the end of the run, after the steel is hot. The object of this is to produce vigorous boiling in the bath, exactly as the rod makes the steel boil, in order to facilitate the removal of oxides from the metal. In other respects the heat is not unusual, though the amount of ore required to eliminate the carbon is higher than in the first four heats. This was due largely to the higher silicon and manganese of the charge.

In heat (6), a great deal of ore was necessary to eliminate the carbon. In this heat, the steel melted high in carbon and vigorous measures were necessary to get things going. A heat made soon after this one, in the same furnace, contained a great deal of foundry scrap (high in silicon), lay flat and not boiling when melted (because the silicon was still present in considerable quantity and protected the carbon from oxidation), and though the steel melted at only 1.25 per cent. carbon, 2200 lb. of ore were required to bring the carbon to .19 per cent. Where much steel foundry scrap, with its comparatively high silicon, has to be melted, this condition will be faced daily unless provision is made for the high silicon in the scrap by using low silicon pig, or less pig than usual, as already described.

(1) 15-TON ACID FURNACE—PRODUCER GAS—CASTINGS CHARGE

| Low phosphorus pig | 6,000 lb. |
|--------------------|------------|
| Billets | 12,700 lb. |
| Scrap | 11,300 lb. |
| | |
| D | 30,000 lb. |

| | С | Mn | Si | P | S |
|-----------|------|---------|----------|------|-------|
| Charge | I.I2 | .90 | .39 | .035 | .034 |
| Aimed for | . 40 | .65/.70 | . 28/.32 | .04 | .04 |
| Obtained | .40 | .75 | .33 | ,026 | . 033 |

| Carbon estimated | .35 |
|------------------|-------|
| Carbon added | . 085 |
| Manganese added | .73 |
| Silicon added | |

| Material | Pounds | C lb. | Mn lb. | Si lb. |
|--------------|-------------------|-----------------------|--------|-------------|
| Ferrosilicon | 200 265 100 | 4.24 16.59 4.00 | 212.0 | 1.33 .80 |
| | | 24.83 | 212.4 | 102.13 |

HISTORY OF HEAT

| Time | No. test | Item | Carbon |
|-------------------|----------|------------------------------|----------|
| 5.30 to 6.50 p. m | | Charged | |
| 2.00 a. m | | Melted | |
| 2.05 a. m | (1) | R 1" | |
| 2.20 a. m | | 100 lb. limestone | |
| 2.40 a. m | (2) | R 2" | - 55 |
| 3.10 a. m | (3) | R 2" | . 44 |
| 3.40 a. m | (4) | R 2" | |
| 3.55 a. m | | 700 lb. pig | |
| 4.15 a. m | (5) | R 2" | . 38 |
| 4.45 a. m | BR | R 2" | .34 |
| 4.50 a. m | | 100 lb. pig | |
| 5.00 a. m | | Tapped | |
| | | 200 lb. ferrosilicon, 50% | ! 1 J1. |
| | | 265 lb. ferromanganese, 80% | in ladle |
| | | Slag light brown throughout. | |

(2) 15-TON ACID FURNACE—PRODUCER GAS—CASTINGS CHARGE

| Low phosphorus pig | |
|--------------------|------------|
| Billets | 24,000 lb. |
| | |
| Rase | 30,000 lb. |

| | С | Mn | Si | P | S |
|-----------|------|---------|---------|------|------|
| Charge | I.22 | .86 | .46 | .042 | .036 |
| Aimed for | . 25 | .65/.70 | .28/.32 | .05 | .05 |
| Obtained | . 27 | .63 | .32 | .033 | .039 |

| Carbon estimated | . 24 |
|------------------|------|
| Carbon added | .073 |
| Manganese added | .744 |
| Silicon added | . 37 |

FIGURED AS FOLLOWS

| Material | Pounds | C lb. | Mn lb. | Si lb. |
|----------------|--------|--------|--------|----------------|
| Ferrosilicon | 210 | 4 · 45 | | |
| Ferromanganese | 270 | 16.90 | 216 | 1.35 |
| | | 21.35 | 216 | 106.35 |
| | | .073% | .744% | 106.35 -37% |

HISTORY OF HEAT

| | | · · · - · | | |
|-------------------|-----|----------------------|----------------------|--------|
| Time | | Time No. test Item | | Carbon |
| 1.30 to 2.50 a. m | | Charged | | |
| 12.00 Noon | | | | |
| 12.05 p. m | (1) | | Brown | |
| 12.10 p. m | | | | |
| 12.45 p. m | | 200 lb. ore | | |
| 1.05 p. m | (2) | R 3" | Greenish brown) | |
| | | | thick] | . 55 |
| 1.10 p. m | | 5 shovels loam | | |
| 1.35 p. m | (3) | R 3" | Yellowish brown. | .47 |
| 2.05 p. m | (4) | R 2" | Yellowish green | . 36 |
| 2.55 p. m | BR | R 1" | Yellowish green | . 25 |
| 3.00 p. m | | Tapped | | |
| | | 210 lb. ferrosilicor | | la. |
| | | 270 lb. ferromang | anese, 80% 🖯 III lau | 16 |

(3) 40-TON ACID FURNACE—PRODUCER GAS—CASTINGS CHARGE

| Low phosphorus pig | |
|--------------------|------------|
| Scrap billets | 13,300 lb. |
| Scrap | 19,700 lb. |
| Turnings | 7,900 lb. |
| Scrap ingots | 27,900 lb. |
| | |

86,000 lb.

Base...... 84,000 lb.

| | С | Mn | Si | P | S |
|-----------|------|---------|---------|------|------|
| Charge | 1.22 | .86 | .46 | .042 | .036 |
| Aimed for | .25 | .65/.70 | .28/.32 | .05 | .05 |
| Obtained | . 26 | . 69 | . 32 | .032 | .038 |

| Carbon estimated | . 09 |
|------------------|------|
| Carbon added | .157 |
| Manganese added | .689 |
| Silicon added | 303 |

FIGURED AS FOLLOWS

| Material | Pounds | C lb. | Mn lb. | Si lb. |
|----------------|--------|--------|--------|---------|
| Ferromanganese | 650 | 40.69 | 520 | 3 . 25 |
| Coal | 100 | 50.00 | | |
| Ferrosilicon | 2820 | 41.17 | 59.22 | 327.12 |
| | | 131.86 | 579.22 | 330.37 |
| | | .157% | .689% | .393% |

HISTORY OF HEAT

| Time | No. test | Item | Carbon |
|--------------------|----------|---------------------------------|------------------|
| 9.55 to 10.25 p. m | | Charged | |
| 7.50 a. m | | Melted | |
| 7.55 a. m | (1) | R 2" | .65 |
| Several lots | | 1000 lb. ore | . |
| 9.10 a. m | (2) | R 3" | .37 |
| 10.10 a. m | (3) | R 2" | . 26 |
| 12.10 p. m | (4) | R 2" | .12 |
| 1.10 p. m | (5) | R 2" | .09 |
| 1.45 p. m | | 1000 lb. pig | |
| 2.10 p. m | (6) | R 2" | . 11 |
| 2.45 p. m | BR | | .10 |
| 2.50 p. m | | 2820 lb. 12% ferrosilicon (hot) | |
| 3.05 p. m | 1 | Tapped | |
| | | 650 lb. ferroman- | |
| | | ganese (80%) in ladle | |
| | | 100 lb. hard coal | . <u></u> |

(4) 15-TON ACID FURNACE—PRODUCER GAS—NICKEL STEEL CHARGE

| Low phosphorus pig | 4,800 lb. |
|--------------------|------------|
| Washed metal | 3,600 lb. |
| Nickel-steel scrap | 14,700 lb. |
| Scrap billets | |
| Nickel plaquettes | 356 lb. |

24,356 lb.

Base.....24,000 lb.

| | С | Mn | Si | P | S | Ni |
|-----------|---------|---------|---------|------|------|------|
| Charge | 1.60 | .79 | .33 | .028 | .027 | |
| Aimed for | .38/.42 | .65/.75 | .14/.18 | .03 | .03 | 3.50 |
| Obtained | . 38 | .75 | . 1 1 | .024 | .04 | 3.29 |

| Carbon estimated | . 28 |
|------------------|-------|
| Carbon added | . 150 |
| Manganese added | |
| Silicon added | |
| Nickel melted | 2.44 |
| Nickel added | |

FIGURED AS FOLLOWS

| Material | Pounds | C lb. | Mn lb. | Si lb. |
|--------------|-------------------|------------------------|------------------------|-----------------------|
| Ferrosilicon | 310 285 350 | 4.52 17.84 16.00 | 6.51 228.00 7.00 | 35.96 1.43 5.25 |
| | | 38.36 .159% | 241.51 | 42.64 |

HISTORY OF HEAT

| Time | No. test | Item | Ni | C |
|-------------------|-----------------|---------------------------|----------|----------|
| 6.45 to 7.40 p. m | | Charged | | |
| 1.30 a. m | | Melted | <i>.</i> | |
| 1.35 a. m | (1) | R 1" | 2.44 | 1.12 |
| 1.50 a. m | | 200 lb. ore | | <i>.</i> |
| 2.15 a. m | | 200 lb. ore | | |
| 2.45 a. m | (2) | R 1" | | - 55 |
| 3.00 a. m | | 100 lb. limestone | | |
| 3.15 a. m | (3) | R 3" | | .41 |
| 3.45 a. m | (4) | R 2" | | -33 |
| 4.15 a. m | BR | | | . 25 |
| 4.16 a. m | | 262 lb. Ni plaquettes | | |
| | | 310 lb. ferrosilicon(12%) | | |
| 4 07 0 77 | | 350 lb. pig | | |
| 4.27 a. m | • • • • • • • • | 285 lb. ferromanganese | | |
| | | (80%) | | |
| 4.35 a. m | | Tapped | | |

Slag heavy—dark brown at first, then light greenish gray.

(5) 15-TON ACID FURNACE—PRODUCER GAS—FORGING INGOTS CHARGE

| Low phosphorus pig | |
|--------------------|-----------------------|
| Billets on floor | 23,500 lb. 500 lb. |

| | С | Mn | Si | P | S |
|-----------|---------|---------|---------|------|------|
| Charged | 1.38 | 1.03 | - 53 | .035 | .047 |
| Aimed for | .30/.35 | .60/.70 | .14/.18 | .045 | |
| Obtained | .34 | .64 | .14 | .032 | .047 |

| Carbon estimated | . 27 |
|------------------|------|
| Carbon added | .075 |
| Manganese added | .71 |
| Silicon added | . т8 |

FIGURED AS FOLLOWS

| Material | Pounds | C lb. | Si lb. | Mn lb. |
|--------------|------------|---------------|-------------|----------------|
| Ferrosilicon | 350 195 | 5.11 12.21 | 40.6 .98 | 7·35 156.00 |
| | | 17.32 | 41.58 | 163.35 |

HISTORY OF HEAT

| Time | No. test | Item | Carbon |
|-------------------|----------|---|--------|
| 6.20 to 7.30 a. m | | Charged | |
| 1.15 p. m | | l = | |
| 1.20 p. m | (1) | Quiet | |
| 1.25 p. m | | 400 lb. ore | |
| 1.55 p. m | | 200 lb. ore | |
| 2.25 p. m | | 200 lb. ore | |
| 2.45 p. m | (2) | R 3" | .93 |
| 2.50 p. m | | 200 lb. ore | |
| 3.15 p. m | (3) | R 3" | .82 |
| 3.20 p. m: | | roo lb. ore | |
| 3.45 p. m | (4) | R 2" | - 59 |
| 4.15 p. m | (5) | R 2" | |
| 4.45 p. m | . (6) | R 2" | |
| 4.50 p. m | | 500 lb. billets | |
| 5.15 p. m | BR | • | |
| 5.25 p. m | | 350 lb. ferrosilicon (11%) | |
| 5.35 p. m | | Tapped | |
| | | 195 lb. ferromanganese in ladle | |

Slag light brown, greenish at end.

| (5) | 15-TON | ACID | Furi | NACEI | PRODUC | CER | Gas-Fo | RGING | INGOTS | |
|-----|----------|--------|-------|-------|--------|-----|--------|-------|--------|--|
| | | | | CF | HARGE | C | | | | |
| L | ow phosi | ohorus | s pig | | | | | . 4 | 800 lh | |

| Low phosphorus pig | |
|--------------------|------------|
| Washed metal | |
| Scrap | 15,100 lb. |

23,500 lb.

Base..... 23,000 lb.

| | C | Μ'n | Si | P | S |
|----------|-----------------|----------------|---------|------|------|
| Charge | 1.60 .24/.28 | ·79 .60/.65 | ·33 | .028 | .027 |
| Obtained | .24/.20 | .61 | .14/.10 | .020 | .035 |

Carbon estimated17Carbon added088Manganese added716Silicon added185

FIGURED AS FOLLOWS

| Material | Pounds | C lb. | Mn lb. | Si lb. |
|--------------------|-----------|--------------|--------|--------------|
| Ferrosilicon | 350 75 | 5.11 3.00 | 7·35 | 40.6 1.13 |
| Pig Ferromanganese | | 12.21 | 156.00 | 0.98 |
| | | 20.32 | 164.85 | 42.71 |
| | | .088% | .716% | .185% |

HISTORY OF HEAT

| Time | No. test | Item | Carbon |
|-------------------|----------|--|----------------------------|
| 6.00 to 6.50 p. m | | Charged | |
| 1.00 a. m | | Melted | |
| 1.05 a. m | (1) | Quiet | Test too hard to drill. |
| 1.30 a. m | | 700 lb. ore | |
| 2.00 a. m | | 300 lb. ore | |
| 2.40 a. m | | 200 lb. ore | |
| 3.15 a. m | | 200 lb. ore | |
| 4.00 a. m | | R 4" | .62 |
| 4.30 a. m | | R 3" | . 50 |
| 4.35 a. m | 1 | 300 lb. ore | |
| 5.00 a. m | | R 3" | ì |
| 5.30 a. m | 1 270 | R 2" | . 27 |
| 6.00 a. m | (6) | R 2", | . 21 |
| 6.30 a. m | BR | | .15 |
| 6.37 a. m | | 75 lb. pig. | |
| 6.50 a. m | | Tapped | |
| | | 195 lb. ferromangenese (80%) in ladle. | |

The Basic Process. Melting.—The charge in the basic furnace generally contains more pig iron (and cast iron scrap) than is used in the acid process, and up to 50 per cent. pig is often charged in regular pig and scrap practice. It is not at all necessary to use such heavy percentages of pig, more especially when soft steel is being melted, since such heats frequently melt very high in carbon, and consequently require considerable amounts of ore. As in the acid process, it is best if possible to charge such a mixture that the bath when melted shall contain I per cent. carbon or less, and very little silicon or manganese. The latter, however, seldom goes below .20 per cent. in basic practice. In order to secure this analysis in the molten bath, the carbon in the charge should be about 1.50 per cent., manganese about 1 per cent., and silicon .50 to .60 per The use of about 1 per cent. manganese in the charge in basic practice is highly desirable, for the reasons already set forth in discussing acid steel making. The manganese seems to be essential to the production of fine steel in both Bessemer and open-hearth practice. The examples of basic heats shown a little further on illustrate several typical charges, and it will be noted that in some cases a very high carbon charge will melt quite low in carbon.

The limestone charged commonly amounts to from 10 to 15 per cent. of the weight of the metal. As in acid practice, the ore consumption should not exceed 6 per cent. of the weight of the steel.

In basic practice it is essential, in order to effect the elimination of phosphorus and sulphur, that a highly basic slag be formed from the start. The limestone is, therefore, charged first. On top of the stone is placed the scrap, and the pig is put in last. Because the bottom is basic, neither the limestone nor the cinder from the melting scrap can attack it and form slag, so that this method of charging is quite safe and enables us to secure a basic slag as soon as melting starts. Moreover, the scrap is protected from excessive oxidation by means of the carbon, silicon and manganese of the pig iron, which melts first and soon partially buries the scrap. There is only one drawback to this order of charging, which is that when much light scrap, especially turnings, is melted, it sticks to the bottom and makes a good deal of work for the men, who are obliged to poke it loose with bars.

Iron ore, or mill scale, is sometimes charged with the limestone, especially when a high percentage of pig iron is melted, in order that as much silicon and manganese as possible may be eliminated

during the melting. They, of course, cannot attack the bottom, for the same reason that the cinder cannot do so.

A basic heat, when melted, unless very hot, is commonly covered with a blanket of sluggish slag, in which many lumps of limestone float about. As the temperature rises, the stone melts, because the slag is able to absorb more and more lime (CaO) as it grows hotter, since the melting point of these slags is the higher the greater the proportion of bases to acid; and therefore the hotter the slag, the more lime it can absorb. When the heat is "down," the helper rabbles up all scrap that sticks to the bottom, and when his metal is all melted takes the first test, as in acid practice.

If the slag formed on the melted bath is very basic and sluggish, as described above, the phosphorus in the steel generally will be largely eliminated when the bath is melted, and the efforts of the men must be devoted to bringing down the carbon of the bath by additions of ore without making the slag too irretentive of phosphorus. The fracture of the test piece and, in shops where very close control of analysis is desired, a quick analysis of the test, show the amount of carbon and of phosphorus that have to be eliminated, and the additions of iron ore and lime are proportioned accordingly.

Before any ore is added, the slag should be nearly all melted. Though the ore assists in promoting the fluidity of the slag, additions of fluorspar may be needed to reduce its melting point, and enable the last of the limestone to melt. In case the slag is very thin, watery and black, indicating an excess of silica and iron oxide, and a deficiency of CaO, burned lime (or occasionally limestone) is shoveled in, to bring the composition and consistency to the desired point. Such a watery slag indicates that dephosphorizing during melting has been less complete than is desired, and extra pains must be taken to secure a limey slag.

Ore is added in sufficient amounts to oxidize the carbon and produces a vigorous boil, as in acid practice. When its effect begins to wear off, and the carbon approaches the desired point, additions of lime are made to keep the slag as basic (and sticky) as possible, in order to promote the elimination of phosphorus. The progress of this elimination is watched closely, and in many cases it is sufficiently advanced by the time the carbon is low enough, and the heat hot enough, to tap. If very low phosphorus content is desired, however, especially if the charge is very high in that impurity, the bath may have to be brought down to .10 per cent. carbon or

lower, and heavy additions of lime and fluorspar made. Meanwhile the carbon of the bath may, if desired, be kept up by additions of pig iron. This is the more to be recommended, as by the boiling produced it aids in keeping the bath hot and in bringing all the steel in contact with the dephosphorizing slag.

The elimination of sulphur is very erratic in the basic furnace, as the examples shown later well illustrate, and though in a general way it follows that of phosphorus, yet in some cases the sulphur can be eliminated to the desired degree only by bringing the bath to very low percentages of carbon and making heavy additions of lime. It will be noticed in these typical heats that the total amount of sulphur eliminated is much less than that of phosphorus.

The same precautions that have been described in the case of the acid furnace should be observed in basic practice. The ore should be got in early, and allowed to work until its iron oxide has been used up as far as possible in eliminating silicon, manganese, and carbon from the bath. By working heats for some time after the effect of the ore has worn off, the iron oxide is as far as possible eliminated from both slag and steel, resulting in the best possible metal. No heat should be tapped that is highly oxidized and "wild," but the process continued until the tests pour quite clean from the spoon. It is possible, and even in some cases common practice, to rush a heat, running the carbon down rapidly with ore, and stopping its elimination at the desired point with pig iron or ferrosilicon, but steel so made is of inferior quality. This is the more true because the basic slag is at the best rather high in iron oxide, and hence oxidizing, to the last.

In basic practice, the recarburizers, especially the manganese, are added largely in the ladle. One reason for this is that manganese, coming in contact with the slag, reduces more or less of the calcium phosphate and causes rephosphorizing of the steel; another is that much more of the manganese added to a basic furnace is lost by oxidation than in acid practice. However, in the manufacture of basic steel for very high-grade forging work, in one of the leading American shops, it is customary to add a part at least of the manganese, in the furnace. The alloy is charged in heavy chunks which sink through the slag and as it is largely immersed in the steel while melting it does not cause marked rephosphorizing, while its effect in eliminating the oxides from the bath and improving the quality of the steel is very marked.

The pig iron, if used solid, must be added in the furnace; coal or

coke is often used in the ladle, though as a rule the best practice demands that as much of the carbon as possible be added in the metallic recarburizers. Ferrosilicon is generally added in the ladle.

In figuring recarburizers for basic heats, allowance has to be made for the considerable proportion of manganese, generally from .20 to .40 per cent., retained by the steel. The examples here given illustrate this point very well.

Typical Heats.—These typical heats were made in large furnaces for high-grade forging work, and like the examples of acid heats already given, consumed much more time than would be taken to-day; they serve very well, however, to illustrate basic practice for the production of steel very low in phosphorus and sulphur. In all of them rather low phosphorus charges were melted, to make sure of attaining the great purity desired, and for the same reason the working of the heat was carried further than usual foundry practice requires. A special basic pig, containing only .3 per cent. phosphorus, was used. Ordinarily, basic pigs contain from .6 to 1 per cent., and the charge from .3 to .5 per cent. phosphorus.

Heat (1) was originally charged for making .48 per cent. carbon steel, and afterward changed over to .30 per cent. carbon. As it melted very high in carbon, 3000 lb. of ore were needed to run the carbon down, and a great deal of lime and fluorspar to reduce the phosphorus, which also melted rather high, considering the requirements and the fact that the limestone charged amounted to 14 per cent. of the weight of the heat. The fluorspar is added to make the very limey slag fluid.

Heat (2) melted much lower in carbon, and pig had to be thrown in to keep the carbon up. The phosphorus was very obstinate after it had reached .03 per cent., and as will be seen, it would not go down to .012 per cent. until the carbon had reached very low figures.

Heat (3) melted very low in phosphorus, and staid down. While waiting for the carbon to reach low enough figures, lime was added to keep the slag basic.

Heat (4) is included to show a heat which melted very high in carbon, but low in phosphorus. The phosphorus content of this charge was purposely made lower than usual because very low phosphorus steel was to be made, with high carbon. Had the charge contained the usual amount of phosphorus, the heat would probably have melted too high in that element, and refused to go down to very

low figures until the carbon had been largely eliminated. The objection to this was that a great amount of pig iron or spiegel would have been necessary as recarburizer to obtain the carbon needed, and this shop had no cupolas at its command for melting recarburizer. The large plants making basic rail steel commonly use hot metal in large amounts as recarburizer, and in this way are able to work high phosphorus charges down low in both carbon and phosphorus and recarburize to high carbon content without trouble.

Heat (5) is included to show a heat that dropped to very low phosphorus content without much working of the slag with lime. Evidently this slag was already very limey and thick, as fluorspar was added after the ore had done much of its work. The amount of limestone charged with this heat was practically the same as that in heat 1, but in this case its effect was pronounced.

Heat (6) melted a good deal like heat (2), though the carbon of the charge was much lower, but the phosphorus, probably because less had to be eliminated, proved much less obstinate, and went down to low figures while the carbon was still comparatively high. Evidently the slag was limey and thick early in this heat, though the amount of limestone used was but 9.4 per cent. of the weight of the charge, as fluorspar was added heavily soon after the bath was melted.

(1) 35-TON BASIC FURNACE—PRODUCER GAS—FORGING INGOTS CHARGE

| Basic pig | |
|-----------------------|--------------|
| High phosphorus scrap | 26,600 lb. |
| Low phosphorus skull | . 11,400 lb. |
| Limontono | 76,000 lb. |

| | С | Mn | Si | P | S |
|----------|-----------------|-----------------|----------------|-------|------|
| Charge | 2.25 .26/.30 | 1.05 .50/.65 | .60 .14/.18 | . 199 | .06 |
| Obtained | . 28 | .73 | . 13 | .029 | .021 |

| Carbon estimated | . 25 |
|------------------|-------|
| Carbon added | . 039 |
| Manganese added | .37 |
| Silicon added | 2 T |

| Material | Pounds | C lb. | Mn lb. | Si lb. |
|-----------------------------|------------|----------------|---------------|----------------|
| Ferrosilicon Ferromanganese | 300 350 | 6.36 | 280.0 | 151.62 |
| | | 28.87 .039% | 280.0 ·37% | 153.39 .21% |

HISTORY OF HEAT

| Time | No. test | Item | C | I P | Mn |
|-------------------|----------|---|-------------------------|------|---------|
| 8.10 to 9.15 a. m | | Charged | | 1 - | |
| 5.15 p. m | | Melted hot | | | |
| 5.20 p. m | 1 | Quiet | | .071 | |
| 5.30 p. m | 1 . | 600 lb. ore | | | |
| 6.30 p. m | 1 | 600 lb. ore | | | |
| 7.00 p. m | | R 1" | | 065 | .35 |
| 7.15 p. m | 1 , , | 600 lb. ore | | | .35 |
| 7.30 p. m | | R 1" | | | .34 |
| 7.45 p. m | | 600 lb. ore | | .003 | .34 |
| 8.15 p. m | (4) | R 1" | | | .30 |
| 8.30 p. m | 1 1 | 600 lb. ore | | | .30 |
| 8.55 p. m | (5) | R r" | | .05 | .25 |
| 9.00 p. m | 1 1 | 400 lb. fluorspar | - | | |
| 9.25 p. m | (6) | R 1" | | .05 | .31 |
| 9.50 p. m | | 400 lb. lime | | | |
| 9.55 p. m | (7) | R 1" | | .035 | .34 |
| 10.00 p. m | | 1200 lb. lime | | | |
| 10.25 p. m | (8) | R r" | | .035 | .33 |
| 10.35 p. m | | 600 lb. lime; 400 lb. | | | |
| | | fluorspar. | | | |
| 10.55 p. m | . (9) | R 1" | .45 | .03 | .37 |
| 11.10 p. m | | 400 lb. lime | | | |
| 11.25 p. m | . (10) | · • • • · · · · · · · · · · · · · · · · | .38 | .02 | . 36 |
| 11.45 p. m | | 400 lb. lime | | | |
| τι.55 p. m | 1 | | | .018 | .36 |
| 12.05 a. m | 1 1 | 700 lb. lime | · · · · · · · · · · · · | | |
| 12.25 a. m | . BR | , | . 25 | | |
| 12.35 a. m | | Tapped 300 lb. feri | rosilicon | | |
| | | { (50%). | | } i | n ladle |
| | | 350 lb.ferror | nanganese | | |
| | 1 | (80%). | | J | |

(2) $_{ m 35}\text{-ton}$ Basic Furnace—Producer Gas—Forging Ingots Charge

| Basic pig High phosphorus scrap Low phosphorus skull | 30,400 lb. |
|--|--------------------------|
| Limestone | 76,000 lb. 10,600 lb. |

ANALYSES

| | С | Mn | Si | P | S |
|-----------|-----------|---------|---------|-------|-------|
| Charge | 2.08 | 1.01 | .56 | . 189 | . 064 |
| Aimed for | . 26/. 30 | .60/.65 | .14/.18 | .015 | .025 |
| Obtained | . 29 | .61 | . 16 | .014 | . 035 |

| Carbon estimated | . 10 |
|------------------|------|
| Carbon added | . 22 |
| Manganese added | . 58 |
| Silicon added | . 21 |

FIGURED AS FOLLOWS

| Material | Pounds | C lb. | Mn lb. | Si lb. |
|----------------|--------|--------|--------|--------|
| Ferrosilicon | 300 | 6.36 | | 151.62 |
| Ferromanganese | 490 | 30.67 | 432.0 | 2.45 |
| Coal | 250 | 125.00 | | |
| | | 162.03 | 432.0 | 154.07 |

HISTORY OF HEAT

| Time | No. test | Item | C | P | Mn |
|-------------------|-------------|--------------------------|---------|-------|------------|
| 5.45 to 6.45 a. m | | Charged | | | ĺ |
| | | Melted | | | |
| 3.10 p. m | (1) | | .40 | .03 | . 18 |
| 3.20 p. m | | 300 lb. fluorspar | | | <i>.</i> . |
| 3.40 p. m | (2) | | . 28 | .035 | . 21 |
| 3.50 p. m | | 1000 lb. pig | | | |
| 4.10 p. m | (3) | | . 28 | .035 | . 24 |
| 4.20 p. m | | ∫ 400 lb. lime | | | |
| 4.20 p. m | · · · · · · | 200 lb. fluorspar | | | |
| 4.40 p. m | (4) | | . 24 | .03 | . 28 |
| 4.50 p. m | | 300 lb. lime | | | |
| 5.10 p. m | (5) | | . 15 | .028 | . 28 |
| 5.20 p. m | | 300 lb. lime | | | |
| 5.40 p. m | (6) | | .09 | .013 | . 26 |
| 5.50 p. m | | ∫ 250 lb. lime | | | |
| 3.30 p. m | | \ 1000 lb. pig | | | |
| 6.20 p. m | (7) | | .II | .012 | . 22 |
| 6.30 p. m | | 450 lb. pig | | | |
| 6.50 p. m | BR | R 2" | . 09 | | |
| 7.05 p. m | | Tapped | | ا | |
| | | 300 lb. ferrosilicon (50 | | 1 | |
| | | 490 lb. ferromanganese | e (8o%) | in la | dle |
| | | 250 lb. coal | | J | |

(3) 35-TON BASIC FURNACE—PRODUCER GAS—FORGING INGOTS CHARGE

| Basic pig High phosphorus scrap Low phosphorus skull | 30,400 | lb. |
|--|--------|-----|
| | | |

 76,000 lb.

 Limestone.
 7,600 lb.

 Base.
 74,000 lb.

| | С | Mn | Si | P | S |
|-----------|------|---------|---------|-------|-------|
| Charge | 2.08 | 1.01 | . 56 | . 189 | . 064 |
| Aimed for | | .60/.65 | .14/.18 | .015 | .025 |
| Obtained | . 29 | .71 | . 16 | .022 | .025 |

| Carbon estimated | . 22 |
|------------------|-------|
| Carbon added | . 087 |
| Manganese added | .50 |
| Silicon added | τo |

| Material | Pounds | C lb. | Mn lb. | Si lb. |
|--------------|------------------|------------------------|---------------|----------------|
| Ferrosilicon | 280 455 60 | 5.93 28.48 30.00 | 364.0 | 141.51 2.28 |
| | | 64.41 .087% | 364.0 .50% | 143.79 .19% |

HISTORY OF HEAT

| Time | No. test | Item | C | P | Mn | S |
|---------------------|-------------|---------------------------|------------|-----------|-------|----------|
| 3.20 to 4.20 p. m | | Charged | | | | |
| •••• | | Melted | | | | |
| 12. r 5 a. m | (1) | | .82 | .or | . 25 | . 032 |
| 12.20 a. m | | 500 lb. ore | | | | |
| 1.00 a. m | (2) | | . 53 | .or | . 26 | |
| 1.10 a. m | | { 300 lb. ore | | | | |
| 1.10 | | 200 lb. fluorspar | | | | |
| 1.45 a. m | | | .47 | .01 | . 26 | |
| 2.15 a. m | (4) | | .41 | | | . |
| 2.20 a. m | | 600 lb. lime | | | | |
| 2.45 a. m | (5) | | . 34 | | | |
| 2.55 a. m | | 350 lb. lime | . . | | | |
| 3.15 a. m | (6) | | . 28 |] | | . |
| 3.20 a. m | | 300 lb. lime | | | | |
| 3.55 a. m | ΒR | | . 23 | | | |
| 4. 10 a. m | | Tapped | <i></i> | ا . ب . ا | | |
| | | 280 lb. ferrosilicon (50% |) | 1 | | |
| | | 455 lb. ferromanganese (| 80%) | in | ladle | • |
| | | 60 lb. coal | | J | | |

(4) 35-TON BASIC FURNACE—PRODUCER GAS—SPECIAL FORGING INGOTS CHARGE

| | С | Mn | Si | P | S |
|-----------|---------|---------|---------|-------|------|
| Charge | 2.60 | 1.14 | .83 | . 147 | .053 |
| Aimed for | .80/.95 | .35/.50 | .12/.16 | .015 | . 03 |
| Obtained | .95 | . 38 | . 13 | .012 | .021 |

| Carbon estimated | .87 |
|------------------|------|
| Carbon added | .017 |
| Manganese added | .085 |
| Silicon added | -60 |

| Material | Pounds | C lb. | Mn lb. | Si lb. |
|----------------|--------|-------|--------|--------|
| Ferrosilicon | 175 | 3.71 | | 88.45 |
| Ferrosilicon | 245 | 4.31 | 5.15 | 28.42 |
| Ferromanganese | 70 | 4.38 | 56.00 | .35 |
| | | 12.40 | 61.15 | 117.22 |
| | | .017% | .085% | . 163% |

HISTORY OF HEAT

| Time | No. test | Item | C | P | Mn | S |
|---------------------|----------|----------------------------|---------|------|----------|-------|
| 10.25 to 11.30 p. m | | Charged | | | | |
| ****** | | Melted | | . , | | |
| 6.30 a. m | | 400 lb. fluorspar | | | | |
| 7.00 a. m | (r) | | 2.12 | .012 | . 3 I | . 029 |
| 7. 10 a. m | | 500 lb. ore | | | | |
| 7.30 a. m | | 500 lb. ore | | | | |
| 8.00 a. m | (2) | | 1.62 | .012 | . 22 | |
| 8.15 a. m | | 300 lb. lime | | | <i>.</i> | |
| 8.30 a. m | (3) | | 1.15 | .012 | . 25 | . 023 |
| 8.45 a. m | | 200 lb. lime | | |] | |
| 9.00 a. m | (4) | | 1.03 | | | |
| 9.15 a. m | | ∫ 750 lb. lime | | | | |
| 9.13 a. m | | ∫ 300 lb. fluorspar | | | | |
| 9.35 a. m | | ∫ 350 lb. lime | | | | |
| 9.33 a. 11 | | l 100 lb. fluorspar | | | | |
| 9.50 a. m | ΒR | Rı" | .85 | | | |
| 10.00 a. m | | \mathbf{Tapped} | | | | |
| | J | 175 lb. ferrosilicon (50%) |) |) | | |
| | | 245 lb. ferrosilicon (11%) |) | in | ladle | |
| | | 70 lb. ferromanganese (| 80%) | J | | |

(5) 35-TON BASIC FURNACE—PRODUCER GAS—FORGING INGOTS CHARGE

| Basic pig | 38,000 lb. |
|-----------------------|------------|
| High phosphorus scrap | 26,600 lb. |
| Low phosphorus scrap | 11,400 lb. |

| | 76,000 lb. |
|-----------|------------|
| Limestone | 10,600 lb. |
| Base | 74,000 lb. |

| | С | Mn | Si | P | S |
|-----------|---------|---------|------|------|------|
| Charge | 2.25 | 1.05 | .60 | .199 | . 06 |
| Aimed for | .46/.49 | .50/.55 | .15 | .04 | .04 |
| Obtained | . 52 | .60 | .115 | .022 | .027 |

| Carbon estimated | .44 |
|------------------|-------|
| Carbon added | . 063 |
| Manganese added | .30 |
| Silicon added | |

| Material | Pounds | C lb. | Mn lb. | Si lb. |
|--------------|------------|----------------|--------|---|
| Ferrosilicon | 220 275 | 4.66 | 220 | 111.19 |
| Coal | 50 | 25.00 | | • |
| | | 46.88 .063% | .30% | 112.57 |

HISTORY OF HEAT

| Time | No. test | Item | C | P | Mn |
|-------------------|----------|---------------------------|-------|----------|------|
| 7.35 to 8.40 a. m | | Charged | | | |
| 4.10 p. m | | 400 lb. fluorspar | | | |
| 4.15 p. m | . (1) | | 1.70 | .054 | .49 |
| 4.15 p. m | | 3000 lb. ore (in 1000-lb. | | | |
| | | lots) | | , | |
| 6.35 p. m | | 200 lb. fluorspar | | | |
| 6.40 p. m | . (2) | | . 75 | .012 | .30 |
| 7.10 p. m | . (3) | | . 58 | .012 | .30 |
| 7.50 p. m | . (4) | | . 50 | .012 | . 28 |
| 8.15 p. m | . B R | R 1" | .42 | | |
| 8.20 p. m | | Tapped | | | |
| | | 220 lb. ferrosilicon (50% |)) | 1 | |
| | | 275 lb. ferromanganese | (80%) | in ladle | Э |
| | | 50 lb. coal | , | | |

(6) 35-ton Basic Furnace—Producer Gas—Nickel Forging Ingots Charge

| Basic pig | 21,600 lb. |
|--------------------|------------|
| Cast-iron scrap | 5,000 lb. |
| Nickel-steel scrap | 53,200 lb. |
| Nickel plaquettes | 6ee lh |

80,455 lb.

| Limestone | 7,600 lb. |
|-----------|------------|
| Base | 76,000 lb. |

| | С | Mn | Si | P | S | Ni |
|-----------|----------|---------|-----------|-------|-------|--------------|
| Charge | | | .54 | . 134 | . 043 | |
| Aimed for | . 27/.30 | .60/.70 | . 14/. 18 | -035 | | 3.5 0 |
| Obtained | .31 | .67 | . 15 | .014 | .025 | 3.45 |

| Carbon estimated | . 27 |
|------------------|------|
| Carbon added | .053 |
| Manganese added | |
| Silicon added | . 21 |
| Nickel added | |

| Material | Pounds | C lb. | Mn lb. | Si lb. |
|--------------|-------------------|-----------------------|-----------------|------------------------|
| Ferrosilicon | 175 580 450 | 3.71 8.47 28.17 | 12.18 360.00 | 88.45 67.28 2.25 |
| | | 40.35 | 372.18 .49% | 157.98 |

HISTORY OF HEAT

| Time | No. test | Item | C | P | Mn | Ni |
|--------------------|-------------------|---------------------------|-------|-----------|----------|------|
| 9.20 to 10.20 p. m | | Charged | | | | |
| 6.30 a. m | (r) | | .50 | .029 | . 30 | 2.70 |
| 6.40 a. m | l | 400 lb. fluorspar | | | | |
| 6.50 a. m | | 200 lb. fluorspar | | | <i>.</i> | |
| 7.00 a, m | (2) | | .44 | .014 | . 29 | |
| 7.30 a. m | 1 | | . 36 | .012 | .31 | |
| 7.40 a. m | | 300 lb. lime | | | | |
| 8.00 a. m | (4) | | . 3 T | | | ļ |
| 8.10 a. m | | 400 lb. lime | | | | |
| 8.35 a. m | ВR | Rr" | . 27 | | | |
| 8.40 a. m | . <i>.</i> | 150 lb. lime; 621 lb. | | | | |
| · | 1 | nickel plaquettes. | | | | |
| 8.50 a. m | | Tapped | l | ļ | | |
| • | i | 175 lb. ferrosilicon, 50% | |) | | |
| | | 580 lb. ferrosilicon, 11% | | in ladle. | | |
| | | 450 lb. ferromanganese, 8 | 80% |) | | |

USE OF SMALL FURNACES

Some years ago W. M. Carr built several very small furnaces, of some 2 tons capacity, with a barrel-shaped removable body or hearth which was taken directly to the moulds and poured through a nozzle in the side, which took the place of the tap hole. The slide for raising and lowering the stopper rod was fastened to the side of the furnace. When the metal was ready to pour, a stopper rod was inserted through the charging door, the furnace body picked up with the crane, the end openings closed by means of special doors provided for the purpose, and the steel teemed directly from the furnace into the moulds. Oil was used for fuel, though natural gas

would probably serve as well; it is doubtful if producer gas would work well in such a short furnace.

With this furnace, very light castings were successfully made. Its advantages were that the steel kept hot longer in the furnace than in a ladle, the small size of the heats and the high temperature of the steel, the low cost of installation and the possibility of shutting the furnace down and starting it up again at small expense. The removable feature does not commend itself to the practical man, as it is cumbrous compared to a ladle, and permits the checker work to cool off too much between heats. Moreover, lip pouring is better than pouring with a stopper for light castings, as the stream from a nozzle has too much velocity to permit filling the moulds at the nicely regulated rate so essential to fine work. Again, with this design, re-carburizing would have to be done in the furnace, which would be a disadvantage if the furnace were run basic.

Since this design was introduced, other small furnaces have been installed, of which the McLain-Carter has been the most advertised. These small furnaces, in most cases, burn oil and are operated acid. Though they enable us to pour small castings of open-hearth steel, which can be made considerably better than Bessemer metal, yet they sacrifice some of the advantages of cost of metal that the openhearth furnace has over other processes, because labor and other charges are distributed over so small an output. Moreover, it appears to be true that these small furnaces cannot economically be used for such light castings as are readily poured of Bessemer steel, and that they are at their best only for rather heavier work, which is made sometimes in converter shops, sometimes by openhearth foundries. It appears doubtful if the installation of an open-hearth furnace of less than 5 tons capacity, with the idea of using it for really light work, is justified by the performance of such furnaces as have been built.

Supplying an Electric Furnace.—One of the first electric furnaces built in this country was used to refine open-hearth steel; a single open-hearth furnace supplying this electric furnace with metal. No such installation has ever been tried in steel foundry work, so far as the author is aware, because it has been found impractical to operate an open-hearth furnace on the continuous principle, except when "hot metal" is available; when cold pig and scrap are thrown into an open-hearth bath, even a small proportion of the total capacity of the furnace, the whole charge is so chilled that too much time is consumed in melting the added metal. The advantage

sought in using the open-hearth furnace for the melting would be that the open-hearth furnace ordinarily melts more cheaply than the electric furnace; and it has been found that under these conditions it does not. Of course, a cupola could be used to melt the charges for the open-hearth furnace, but it is difficult to operate a cupola at such a rate as to furnish metal only every 2 hours, as would be necessary in a cupola-open hearth-electric furnace combination. Moreover, the addition of the third unit would seriously complicate the operating schedule.

For the average steel foundry, therefore, it is advisable to operate either open-hearth furnaces or electric furnaces, but not to try to "duplex" and refine open-hearth metal in an electric furnace with the object of getting cheaper electric steel. In shops having a number of open-hearth furnaces, of course, an electric furnace could be supplied with open-hearth metal to make a small tonnage of high-grade steel, because it would be comparatively easy when an open-hearth furnace is being tapped every few hours to arrange the operating schedule of the electric furnace accordingly.

If the open-hearth furnaces were basic, either a basic or an acid electric furnace could be installed; if the acid open-hearth process were in use the basic electric process would be better, because the acid open-hearth metal is generally comparatively high in phosphorus and sulphur, and the impurities would in most cases have to be reduced in the electric furnace if the best possible steel were desired.

THE RAW MATERIALS

The raw materials for the open-hearth furnace may be summarized as follows:

ACID PROCESS

Specified Analysis

| Material | P | S | Si | Mn |
|----------------------|-------------|-------------|------------|------------|
| Low phosphorus pig | .03 to .04 | .03 to .04 | as desired | as desired |
| Low phosphorus scrap | | .03 to .04 | | |
| Washed metal | | .03 to .035 | | |
| Wrought iron | .03 to .035 | .03 to .035 | | |
| Iron ore | | .03 to .035 | | <u> </u> |

BASIC PROCESS

| Basic pig | .60 to 1.00 | .03 to .1 | 1 to 1.25 | |
|-----------|-------------|-----------|-----------|--|
| . 0 | | Ü | | |

Scrap—miscellaneous unclassified cast iron, wrought iron and steel scrap—if the sulphur is too high, production of low sulphur steel will be difficult.

Iron ore low in silica, with phosphorus up to .5 per cent. Limestone low in silica.

The basic furnace of course does not need material low in phosphorus and sulphur, as these are eliminated in the process. The sulphur, however, can be only partially removed, so that the sulphur of the raw material cannot be as high as the phosphorus. The silicon, too, must be low, since limestone must be provided to flux the SiO₂ resulting from its oxidation, and therefore the more silicon there is in the bath the more lime is used up in fluxing it. For this reason, machine cast pig is preferable to sand cast, as it is free from adhering SiO₂ (sand), which consumes a good deal of lime. For the same reason, both ore and limestone should be low in silica.

The slight loss of metal in the acid process does not raise the phosphorus and sulphur in the steel enough to need figuring. An oil-fired or natural-gas-fired furnace will not pick up sulphur; 1 but a producer gas-fired furnace will pick up at least .or per cent. sulphur per heat. For this reason, the stock melted must be lower in sulphur than the steel desired, and also the gas coal used in the producers must generally be bought to specifications, coal over 1 per cent. sulphur being unsuited for many acid furnaces making fine steels. The amount of sulphur so picked up varies greatly in different furnaces, depending evidently somewhat upon the distance from producers to furnace; the greater this distance, the less the "pick up."

Using the equations with which we worked before on this head, and assuming that we melt 40 per cent. of our own scrap per heat, and that the pick up amounts to .or per cent. per heat, we can figure the percentage of sulphur that we can allow in our raw materials as follows:

Let X equal the allowable percentage in raw material, including ore: .05 per cent. equal the maximum percentage in our steel;

6000 lb. of pig
8400 lb. of purchased scrap
9600 lb. of our own scrap

be the weight of a charge;

23000 lb. be the weight of metal obtained per heat;

¹ This statement is true only of low-sulphur oils. A furnace fired with the high-sulphur oil obtainable to-day will pick up several points of sulphur.

The additions to the furnace be those of acid heat No. 5 in the list of typical heats already given; then.

.05 =
$$\frac{\frac{\text{Pig Scrap Ore Own scrap Ferrosilicon Whole heat}}{6000 X + 8400 X + 1100 X + 9600 (.05) + 350 (.02) + 24000 (.01)}{23000}$$

whence X = .027 per cent.

If the pick up be .02 per cent., and purchased pig and scrap each contain .03 per cent. we shall get, for the maximum sulphur in the steel,

$$X =$$

$$\frac{14400(.03) + 1100(.03) + 9600X + 350(.02) + 24000(.02)}{23000} = .071\%.$$

If it be found, then, that our furnace picks up .02 per cent. let us see what must be the composition, X, of purchased pig, scrap and ore, in order that our steel shall not exceed .05 per cent.

whence X = .012 per cent.

.05 =
$$\frac{14400X + 1100X + 9600(.05) + 350(.02) + 24000(.02)}{23000}$$

whence X = .012 per cent.

We cannot afford to buy material of this analysis, even if we can get it, so that we must buy coal of lower sulphur content, in order to reduce the "pick up;" use some other fuel, such as low-sulphur oil or natural gas; dispose of all our scrap; or change over to a basic furnace. If we do not remelt our scrap, we calculate as follows:

.05 =
$$\frac{\frac{\text{Pig}}{6000X + 18000X + 350(.02) + 1100X + 24000(.02)}}{23000}$$

whence X = .026 per cent., the content of sulphur that we must have in our raw material in order to keep the sulphur in our steel below .05 per cent.

We can, of course, buy scrap, pig and ore of this analysis; but to be obliged to sell all our own scrap is out of the question.

In some cases, for instance in a plant established in the Pacific coast states, pig iron may be very costly and scrap very cheap. The use of ferrosilicon or ferromanganese to contribute carbon, silicon and manganese to a bath of all scrap, following the expedient suggested for Bessemer practice, is not feasible in open-hearth work, as carbon is also essential. It is easy to show that if the ferroalloys were used in proportions sufficient to give the required carbon

in the bath, the resulting manganese or silicon would be many times too high.

In order to be able to handle a bath of all scrap, it has been proposed by Mr. Lash to use gas coke or other material of that sort, charged on the bottom of the furnace under the scrap. As the latter melts it absorbs the carbon of the coke. The carbonaceous material being under a heavy mass of scrap, is protected from oxidation sufficiently to ensure the absorption by the bath of the greater part of the carbon. Though this process is, or has been, used with success, it has its drawbacks (as any method involving a preponderance of scrap must have), in that there is nothing to protect the iron from oxidation in melting. This is not an insuperable obstacle, as in acid practice a good deal of loss from this cause always occurs, but it should not be lost sight of in figuring on the use of the process.

A possible solution of this problem, in cases where pig iron is at a prohibitive price, would be to melt the scrap, or some of it, in a cupola, and thence run it into the open-hearth furnace. A great deal of carbon would be absorbed by the scrap in the cupola, probably from 2.50 to 3.50 per cent., and by pouring this cupola metal upon a partially melted charge of scrap in the open-hearth furnace, a bath would be obtained high enough in carbon to be worked in the usual manner.

Because the cupola could not be run continuously, it would of course not melt as cheaply as in regular cupola practice. Then, too, high proportions of scrap are far from easy to melt in a cupola. But the advantages secured might more than offset the drawbacks.

High-carbon and Alloy Steels.—The manufacture of high-carbon and alloy steels in open-hearth furnaces presents some special features that should be considered.

For the production of high-carbon steels in the acid furnace the method followed is to interrupt operations when the carbon of the bath is just below the content desired. This practice presents no special features, except that the bath must be of high carbon when melted, in order that it may be hot by the time it reaches the composition desired.

In the basic furnace, two methods are open to the steel maker. The first, of which one example is given in basic heat number 4 above, is to operate as in the acid furnace, tapping the heat when the carbon is just below the desired amount. To carry this method out successfully, the charge of the furnace should be comparatively

low in phosphorus and sulphur, because, as we have already seen, it is not always easy to eliminate these impurities to the desired degree during the working of a bath that is kept high in carbon. The second method, which is more often used with charges high in phosphorus and sulphur, is to carry the carbon down below .10 per cent., in order to promote the elimination of the phosphorus and sulphur. This necessarily involves a very heavy addition of recarburizer, since to attempt to use coal or coke to bring heats of steel up from .08 per cent. to as high as .70 per cent. carbon, will almost inevitably result in the turning out of many heats of incorrect analysis. The ladle additions of ferromanganese and ferrosilicon will not contain nearly enough carbon to raise the carbon of the bath to the desired point, and we are thus forced to use pig-iron additions. For a 30-ton heat (say 66,000 lb.), of basic steel, containing say .10 per cent. carbon, to be brought to .80 per cent. carbon, .40 per cent. silicon, and .70 per cent. manganese, our additions will be about as follows:

| | C per cent. | Si per cent. | Mn per cent. |
|-------------------|-------------|--------------|--------------|
| Ferromanganese at | 6.00 | - 5 | 80.0 |
| Pig at | 4.00 | 2.0 | · 5 |
| Bath at | .10 | .05 | . 20 |

| Item | Wght., lb. | C lb. | Si lb. | Mn lb. |
|----------------|------------|---------------|--------|--------|
| Pig | 11,300 | 452 | 226 | 56.5 |
| Ferromanganese | 370 | 22.2 | 1.85 | 296 |
| Bath | 54,330 | 54 · 3 | 27.17 | 108.7 |
| | 66,000 | 528.5 .80% | 255.02 | 461.2 |
| | | .80% | .39% | . 70% |

Several considerations affecting the use of these heavy pig-iron additions present themselves immediately. To begin with, to add some II,000 lb. of solid metal to a 54,000-lb. heat, will chill the bath to such a point that it will have to be heated up all over again, which can only be done by boiling out a large part of the carbon already added. It is essential, therefore, that the pig be added molten, and cupolas will be needed to melt it.

Moreover, if we add such quantities of molten basic pig, of the usual phosphorus content of at least .60 per cent., we shall raise the phosphorus of our steel to a prohibitive degree, as a little figuring will show:

Even if a special low phosphorus basic pig (phosphorus .30 per cent.) is used, the phosphorus of the steel will be too high.

or .062 per cent. phosphorus.

Therefore, we are forced to use Bessemer or low phosphorus pig as recarburizer, in this method of working. The resulting content of phosphorus in the steel will be as follows:

54,000 lb. soft steel at. 015 per cent. phosphorus....... 8.1 lb. 11,000 lb. Bessemer pig at .10 per cent. phosphorus..... 11.0 lb.

19.1 lb. or .020 per cent. phosphorus.

54,000 lb. soft steel at .015 per cent. phosphorus....... 8. ī lb. 11,000 lb. low phosphorus pig at .03 per cent. phosphorus. 3.3 lb.

11.4 lb. or .017 per cent. phosphorus.

Of the alloy steels, nickel steel can be readily produced in an openhearth furnace, either acid or basic. The nickel is added in the form of plaquettes charged with the heat or thrown into the bath, and there is almost no loss in melting; if nickel-steel scrap is available, a considerable proportion of it can be remelted in either the acid or the basic furnace with but slight loss of nickel. If very high percentage nickel steel is to be made, so that a heavy addition of nickel plaquettes is necessary, the nickel will have to be added in several lots. While it is melting, the bath tends to lose carbon. It is usual in this case to begin adding the nickel when the carbon of the bath is somewhat above the desired content; and in the acid furnace, the silicon is generally charged before the nickel additions are begun, to prevent this elimination of carbon as far as possible.

Other alloys, such as chrome, tungsten, etc., are oxidized almost completely in remelting the scrap. Chrome in the basic furnace, starting at 1 per cent., will generally go down to .10 per cent. by the time the heat is ready to tap. Moreover, the chromic acid resulting from the oxidation of the chrome produces a very pasty

slag, which, especially in the basic furnace, makes trouble by "building up" on the bottom to a serious extent. Plants remelting chrome steel scrap as a large part of their charge generally have to run one or two heats of carbon steel every little while to cut out these slag accretions.

Ferrochrome of 65 per cent. chrome should be added in the furnace and, at least in basic furnaces, a considerable proportion of the chrome is oxidized in melting. It has been found that the amount of chrome so entering the slag as oxide is greatest immediately after melting, and that by holding the heat from 20 minutes to ½ hour after the ferrochrome is melted, quite a considerable amount of the lost chrome will be reduced by the carbon of the bath and restored to the steel. The French metallurgists assist this action by spreading coke dust over the slag in the manner that has been described in the discussion of the French methods of deoxidizing acid slags. The low-carbon ferrochromes are said to lose more chrome when thrown into the open-hearth bath than do the ferrochromes of higher carbon content, because the carbon of the latter allov is oxidized in preference to the chrome. Certain manufacturers too, supply a ferrochrome containing from 4 to 5 per cent. silicon, the latter element being purposely present in the alloy to protect the chrome from oxidation, in the same manner as the carbon.

Tungsten, molybdenum, etc., can be added to the bath as ferro alloys, or as metallic powder. In either case, it is generally best to put these expensive alloys in sheet metal cans which are then tossed into the bath. When so charging tungsten, it will sink to the bottom of the furnace, owing to its great weight, and generally nearly $\frac{1}{2}$ hour will be necessary to melt it. If the ferrochrome or ferromanganese have been put in the same cans with the tungsten and carried to the bottom of the furnace, the usual loss in adding these alloys to the furnace will not occur, and much smaller additions are made, nearly as small, in the case of manganese, as when adding the alloy in the ladle. Heavy additions of these alloys must be heated to avoid chilling the bath, and will present the same difficulties that are met with in making high percentage nickel steels.

Manganese steel can be made quite easily of open-hearth metal, as of Bessemer metal, and by practically the same methods. In order to avoid heavy losses the large additions of ferromanganese have to be melted and added in the ladle. The scrap produced can be remelted and its iron saved, but in ordinary practice the manganese contained in it is lost. The large heats that have to be

disposed of, and the rather low temperature of the metal compared to Bessemer steel, make the process unsuited to the production of much of the work that the makers of manganese steel castings are called upon to turn out. At least two plants in this country are making manganese steel successfully with open-hearth furnaces. In one of these plants, tilting furnaces of some 15 tons capacity are used, and the steel is made by mixing molten ferro manganese with the proper amount of soft open-hearth steel in the ladle. The furnace can be tilted over far enough before the metal comes out of the tap hole to prevent the slag coming with it (at least until the last ladleful), so that the proper proportioning of the metals is not difficult. The ferromanganese was formerly melted in crucibles, but a cupola is now used.

The other plant makes but a small amount of manganese steel, for which the Bessemer process was formerly used. Of late years, however, they have found it more economical to mix in a ladle molten ferromanganese and open-hearth metal of the usual composition for steel castings, some .20 to .30 per cent. carbon and .60 to .80 per cent. manganese. The open-hearth steel is teemed through a stopper from the big ladle into the small ladle used for the manganese steel. For light castings, it would be almost impossible to keep the metal hot enough were this practice followed.

CHAPTER VI

THE ELECTRIC FURNACE

The electric furnace with basic lining won a place in steel making practice in the first instance primarily because it enabled the steelmaker to carry the elimination of phosphorus, sulphur, and oxides and gases, to a point that cannot be attained with any other steel-making process. This is especially true of sulphur and oxides. The ability to attain such a high degree of purification of the steel is due to the opportunity offered to expose the metal to conditions either oxidizing, neutral or reducing, at the will of the operator, By making an oxidizing slag with iron ore, lime, etc., we can expose our steel to oxidizing conditions. Then, as the atmosphere of the electric furnace can be made almost entirely neutral (or non-oxidizing), by scraping off this slag and making a new one of lime and fluorspar, and using reducing agents such as powdered coke and ferrosilicon, we can produce conditions almost as reducing as we Moreover, the very high temperature attainable enables us to melt slags composed almost wholly of silicate of lime and very free from metallic oxides, which deoxidize and desulphurize the steel almost completely.

The ideal finishing conditions for a heat of steel, a non-oxidizing atmosphere, and a slag (and therefore a steel) practically free from oxides of iron, can only be approximated in the open-hearth furnace. This is true, first, because the flame is always oxidizing, and second, because it is not feasible to melt slags very free from iron oxides and composed almost wholly of silicate of lime, on account of their high melting point. These ideal, but heretofore unattainable, conditions were put at our disposal by the development of the electric refining furnace.

In the last few years the acid-lined electric furnace has been quite widely introduced, and in spite of the fact that it requires more expert supervision than the basic furnace, many foundries have adopted it. As will be explained in detail in later paragraphs, no phosphorus or sulphur can be removed from the steel in the acid process, and the elimination of oxide of iron from the slag cannot be

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carried nearly as far as in the basic furnace. The steel, however, when made under competent supervision compares well with basic electric furnace steel; and as the daily tonnage produced is generally higher than in the basic process, the steel is often cheaper, in spite of the higher cost of the low phosphorus and sulphur scrap necessary for the acid process.

There are no limitations to the electric process from the point of view of the kinds of steel that can be made. We have seen that the crucible process is not strikingly adapted to the manufacture of low carbon steels; that the Bessemer process, on account of the fact that all carbon is eliminated, and the desired amounts restored in the shape of ferromanganese, pig iron, etc., demands fluid additions for high carbon and high alloy steels, and will not make certain steels easily; and that the same is true of the basic open-hearth process when very low sulphur and phosphorus are to be attained, since in this case we frequently have to eliminate nearly all the carbon before we can get the impurities down to the desired point. In the electric furnace, on the other hand, especially the basic furnace, additions of alloys of any kind, in any desired amount, can be made to the furnace, the alloys melted, and the whole bath brought up to high temperature, not only without change in the composition of the bath to which the allovs are added, but even without great loss of the allovs. This is owing to the fact already mentioned, that we can hold our steel for a considerable period in a neutral atmosphere, under a slag that does not act on the carbon, silicon, manganese, or added alloys of the metal.

Still another advantage of the electric furnace has already been mentioned in discussing manganese steel, and is to be inferred from the previous paragraph. This is its ability to melt steel scrap containing costly alloys, without the oxidation and loss of the alloys that occurs in working up the scrap by Bessemer or open-hearth methods, or the great gain in carbon so difficult to avoid in crucible melting. None of these troubles occur with the electric furnace; we can remelt our scrap practically without change of composition, correct the trifling change that does occur with suitable additions, and pour our steel. Manganese steel scrap, for reasons that will be given later, can be remelted only in a basic electric furnace; and the basic furnace is better suited to remelting scrap of certain other alloy steels than is the acid furnace.

Summarizing the sunny side of the question; the electric furnace gives us metal of unsurpassed quality, which is very hot, and will

run the lightest sections with ease. The furnace can be run intermittently as it is not essential to its operation that it be kept continuously hot; but the cost of installation, especially when a source of power must be provided, and the consequent overhead expense, make idle hours an expensive luxury. The process is flexible when small units are used, as has already been explained at length in the introductory chapters; in large installations flexibility is attained only at a sacrifice of some of the excellencies of the process. Steel of any sort can be produced at will.

The electric furnace when first introduced was handicapped by several considerations, most of which were due to the fact that the process had been developed by electrical engineers rather than by steel makers. Some furnaces were designed and built in which the electrical efficiency had been developed at the expense of the metallurgical efficiency, resulting in high operating and maintenance costs and vexatious delays in operations. In some designs, for instance, bottoms could not be so made as to last a proper length of time; and in all types of furnace, in the early days, bottoms were improperly set and required frequent renewal.

Like all new processes, too, the electric furnace suffered from a lack of men trained to operate it properly, and even to-day really capable melters are not by any means easy to find. The cost of installation is rather high compared to that of some of the other processes, and if the plant has to furnish its own power, the installation costs of the power house and transmission lines are almost prohibitive.

Then, too, the cost of the steel in the ladle, 10 years ago, was very high compared to that of the other processes of equal daily tonnage, and this at first discouraged the adoption of the process. Improvements in practice soon reduced the costs, and they were still further lowered when it was realized that the extremely pure steel that was almost always made in the early days is not necessary for most classes of steel foundry work. By working for steel of say .03 to .04 per cent. phosphorus and sulphur, the process in either the acid or the basic furnace can be carried out much faster than if very low phosphorus and sulphur contents are sought; and the quality of the steel so made, if the process is properly controlled, is enough better than that of Bessemer or open-hearth metal to justify the use of the electric furnace.

During the last 10 years, and especially the last 6 or 7 years, the electric furnace has made extraordinary progress in this country,

until to-day we lead the world in the output of electric furnace steel. It is just as well to repeat here, however, a matter that has already been touched upon in the preliminary chapters, and which we may well term the abuse of the electric furnace process. Certain types of furnace have been widely introduced which simply cannot be operated for the production of really high-grade steel; and some operators who should have known better, using furnaces of the best types, have been swept away in the race for tonnage and have tended more and more to imitate the practice of their competitors who were using the badly designed furnaces. The process in these furnaces is necessarily little more than "charge, melt and pour," with ladle additions of silicon, manganese and aluminum; and if the analysis is correct and the steel deoxidized to a proper extent, it is accidental rather than intentional. Some of the worst samples of cast steel ever tested in the author's laboratory were made in one of these furnaces, and amply confirmed Mr. Lindemuth's statement "It is easy in the electric furnace to manufacture steel of a quality as poor as the worst product of any other process and the mere fact that steel comes from an electric furnace signifies nothing as to its quality. Superior steel is, therefore, not a natural product of the electric furnace, but the product of a process made possible by the electric furnace."1

Types of Electric Furnaces.—Electric furnaces as applied to steel making may be divided into two classes, determined by the means of generating the heat and applying it to the bath. In the first type, the metal is heated by its resistance to the passage through it of a heavy electric current. These furnaces have all been designed in such a manner that the bath forms the secondary circuit of an electric transformer, and hence have been styled "Induction furnaces."

In the second type, the metal is heated by the electric arc, struck either between electrode and bath, or between electrode and electrode. These are called "Arc Furnaces," and with comparatively few exceptions are the design in use in the steel industry to-day.

So many different designs of electric furnace have been built, and so many special features, mostly patented, applied to the designs, that to mention them all would be almost an impossible task. We shall confine ourselves in this volume to general types

¹Lindemuth, L. B., "The position of the electric furnace in iron and steel metallurgy:" Am. Electrochem. Soc., vol. xxxvii (1920), p. 299.

and general principles, and no attempt will be made to give even a list of the different designs.

Induction Furnaces.—The original induction furnace consisted of a circular trough containing the bath, which formed the secondary of a transformer, a primary winding of copper wire and suitable iron core pieces, the whole constituting a transformer and furnace in one. By passing a high voltage, low amperage current through the primary windings, a current is induced in the metal in the trough, whose voltage will be to that of the primary current as one (the number of turns in the secondary) is to the number of coils in the primary. Thus a low-voltage, high-amperage current is maintained in the trough.

Cooling devices for the primary winding are necessary, and of course suitable covers for the trough, tipping mechanism and spout, etc. In starting the furnace, a ring of iron or steel is placed in the trough in order to make a closed circuit for the current, and when this is melted cold scrap is charged. After one heat is made, by leaving a part of the metal behind after pouring, a ring is left to start the current. In refining molten metal from another furnace, of course, this is not necessary.

A number of modifications of the original design were made in later installations, with a view to improving the electrical efficiency of the furnace, and of providing a working chamber more easily got at than the annular trough. In at least one of these, a central bath connected with two troughs, the whole forming a figure 8, was provided. For a number of excellent reasons, the induction furnaces as a class have failed to win approval in this country. Some of them required low-frequency current, an objectionable feature, as it called for special generators. Then, as the heat is generated in the metal, and contributed to the slag only by conduction, it is impossible to maintain a very basic slag fluid; and for that reason, as we shall see a little later, desulphurizing of impure steel is not readily carried to the point that can be attained with arc furnaces.

Again, owing to the shape of the space containing the bath, it is not feasible to scrape off all of the oxidizing slag used for dephosphorizing, but a considerable amount of this slag has to be left in the furnace. The reducing conditions used to desulphurize the bath, at once reduce phosphorus from the slag left from the dephosphorizing period, so that a considerable rephosphorizing of the metal takes place. Thus it has been found that the degree of purification

that can be attained in these furnaces is not as high as can quite readily be secured with arc furnaces. Then, too, as may readily be imagined, the annular passages and their covers form a most unhandy working chamber, expensive to build and maintain, and most awkward to get at.

Another disadvantage is the circulation of the bath in the narrow annular passages, which results in a mechanical erosion of the lining, in addition to the usual cutting action of the slag. In order to reduce the effect of this motion to a minimum, the annular passages are made of as great cross-section as possible; but there is a very definite and precise limit to the enlargement of the cross-section of the channels, inasmuch as the current density must be kept high in order to produce the requisite heat in the metal.

The worst feature of all, however, is the difficulty if not impossibility, of setting a lining in the induction furnace so that it will "stay put." Brick linings of any kind are unsatisfactory and linings tamped in and afterwards burned in place have proved almost impossible to maintain.

These disadvantages have off-set the one possible advantage of the type, which is that in melting cold stock there are almost no current fluctuations as there are with arc furnaces, so that the load on the power house is steady. Ten years ago this feature claimed attention, but to-day our furnaces are operated with so little trouble from current fluctuations that power companies no longer consider them a disadvantageous load.

It has been claimed, also, that the induction furnace uses less power in melting down than do arc furnaces, but trustworthy figures are difficult to obtain, particularly in view of the failure of induction furnaces of commercial sizes to "make good" in this country. The furnace apparently should be well adapted to melting pure materials that need little refining, but the troubles encountered in attempting to operate induction furnaces, especially bottom troubles, have so far prevented their adoption.

Arc Furnaces.—The arc furnace consists essentially of a melting chamber or hearth containing the bath of metal, and provided with carbon electrodes to introduce the current into the hearth and to carry the arcs. In some of the early types, and in at least two types still in use to a certain extent, the electrodes are brought through the walls of the furnace, either horizontally or at an angle, and the arc is struck between the ends of the electrodes. This design results in heavy wear on the roof, high power costs, and at

least in commercial sizes, constant trouble with the electrodes. The latter break often enough when they are suspended vertically, without inviting trouble by putting them in a position where their own weight tends to bend and break them.

The arc furnaces in general use in this country are provided with vertical electrodes, generally three in number, the arc being struck between the ends of the electrodes and the metal. Alternating current is used, and the three electrode type naturally lends itself well to three phase current, which is the power usually available. Furnaces are built with one and with two electrodes, but these are mostly of small size.

In the simplest type of furnace, which has been by all odds the most successful, the hearth and bottom are used only to hold the steel, and no attempt is made to make them in any way a part of the current-carrying system. Many designs have been tried out in which the bottom is made the common return point for the current, either by using water cooled iron plugs to connect the bath with the framing of the furnace, or by making use of the electrical conductivity of the magnesite bottom when hot. In the latter case iron rods or a carbon block in the bottom next to the shell are provided to act as a collector.

The designers of these types have failed to recognize that a furnace bottom at the best is hard to maintain in good condition, and frequently cuts away at one or more points, forming holes of varying size. The furnaces with water cooled plugs in the bottom gave trouble when, as frequently happened, the bath cut away the bottom around one or more plugs, resulting in some cases in the steel eating away the plug until the water cooling chamber was reached and an explosion followed that resulted in a long shut-down. The same thing, to a lesser degree, is true of all furnaces where the bottom carries current; difficulty with the bottom causes more trouble than in the furnaces where the bottom has nothing to do with the circuit. In some designs, again, the bottom is connected to the third phase of a three-phase system, the electrodes to the other two. These furnaces must necessarily operate for longer or shorter periods when the bottom is cold, on but two phases, with resulting loss of power.

The number of special features that have been tried out, such as auxiliary electrodes to be used in starting up on cold stock and withdrawn afterwards, are too numerous to mention. The experience of 15 years has shown that in this, as in so many things, the

simplest is the best, and the principle object of most, if not all, of these features appears to be to provide a "selling argument" to appeal to the prospective purchaser.

Almost all the arc furnaces are made tilting, which avoids tap hole troubles, and facilitates repairs to the bottom. The furnace is generally round, which gives the best distribution of the heat of the arcs, which is necessarily more localized than the heat source in a

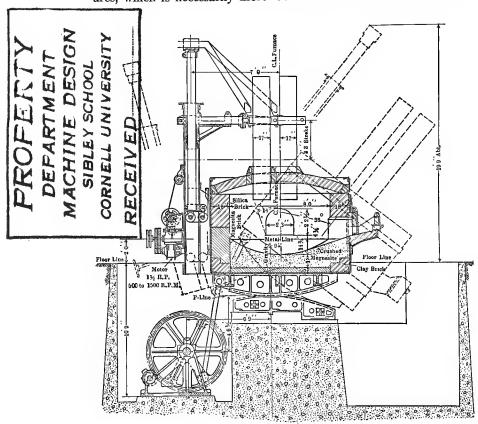


Fig. 11.-Six-ton Heroult furnace.

fuel-heated furnace. The roof is made readily removable, as the roofs require renewal more frequently than the side walls. The roof, in fact, is built in a ring of structural steel quite separate from the rest of the furnace, and a "spare" is always kept on hand, one roof being rebuilt as soon as the other is put on. In some small furnaces

the roof is arranged to take off or tilt up so that the charge may be introduced through the top of the furnace. In large furnaces this practice cannot be successfully carried out, as the joint between roof and walls cannot be kept tight enough when the brick work is partially melted away; and because a roof that is nearly worn out, but still good for several heats if undisturbed, will surely collapse if tilted or lifted off at each heat.

Silica brick are used for roofs in practically all cases, though some makes of clay brick have been tried successfully. Either special blocks of fairly large size, or standard "shapes" are used; the advantage of the former is that a roof built of blocks has fewer joints, where the melting away of the bricks progresses most rapidly. The large blocks, however, are more apt to spall when the furnace cools down, and many furnace men do not use them, especially when operating the furnace "single turn."

In building the roof, a wooden form is used, just as is done in crecting a heating furnace. The silica brick of the roof should be laid dry, with no slurry of any kind in the joints. Silica sand may be spread over the top and brushed into the cracks, but none should be left over the brick. To allow for the expansion of the roof in heating up, wooden liners are placed between the bricks at intervals, the total allowance being about $\frac{1}{16}$ in. per foot; when the furnace is heated, these liners burn out and allow the silica brick to expand. When the roof has once been heated up to working temperature, the bricks are sintered together enough so that none will slip out if the furnace is cooled off.

Once or twice a week, the dust should be blown off the roof of the furnace; a layer of dust or sand even 1/4 in. thick so blankets the radiation of heat as to decrease considerably the life of the brick of the roof.

The electrodes are supported by suitable arms extending over the roof of the furnace, and as they burn away are fed down. Generally, the electrodes can be fed down a total distance of about 6 in. without adjustment; when they are burned off more than this, the water-cooled clamps by which they are grasped are loosened, the electrodes slipped down, and the holder tightened again.

In very small furnaces, the electrodes may be kept in proper position by hand or by hand-operated motors, but for furnaces of commercial size, automatic control is absolutely essential. Several types of these controllers are in use, all of which serve very well to keep the arcs at a fairly constant length, and the current on each

electrode approximately the same, except during the first 10 to 30 minutes in starting up with a cold charge. Then the operator must stand by and supplement the automatic mechanism by raising and lowering the electrodes by the manual controllers that operate the electrode motors.

Even with careful supervision, the current fluctuations in starting up are considerable, as the arc jumps from one piece of scrap to another and frequently "goes out" altogether. Proper charging of the stock minimizes this difficulty, and as most furnaces are fed by power lines carrying many times the demands of the furnace, the central station is not adversely affected. In the case of furnaces operated on their own generator, however, the current fluctuations are a more serious matter, and the generator has to be of special design to endure this trying service.

As arc furnaces all require alternating current at low voltage and high amperage, transformers to step down the current are necessary. They are, of course, always placed close to the furnace, and the current is carried to the electrodes by heavy copper cables and busbars of copper plate. It is desirable to install transformers of ample capacity, as the greatest economies that have been made in electric furnace practice in the last few years have resulted from "driving" the furnace by increasing the current input. On a 3 ton basic furnace under the author's supervision, a load of 900 kw. was at first carried during melting down, the power consumption averaging about 1050 kw.-hr. per ton of steel produced. By increasing the power input to 1100 kw., the consumption of power has been cut to between 700 and 800 kw.-hr. per ton of steel.

In basic practice, one and sometimes two slags are made and scraped off the bath—the round furnaces, therefore, with a door at the spout (and in larger sizes a door at each side also) and no corners that are not easily reached with a rabble, are admirably adopted to good practice. They are, in fact, the type that has been developed by the cooperation of the steel maker and the electrical engineer, and can be maintained and operated with the minimum of labor and expense.

Electrodes.—The electrodes are made either of amorphous carbon or of graphite. The latter can be of smaller cross-section than the carbon electrodes, owing to their lower electrical resistance, but as they are more brittle, full advantage cannot always be taken of this feature. They are made in lengths of from 3 ft. 6 in. to 8 ft. and fastened together by threaded plugs that are screwed into seats

in the electrode ends. When an electrode has so far burned away that it cannot be set any lower in the clamps, a lew length is fastened to it in this manner. The connecting plugs are well daubed with a joint compound furnished by the makers, which serves to improve the electrical conductivity of the joint.

As a chain is no stronger than its weakest link, so an electrode is no stronger than its joints. If the contact between the plug and its seat is imperfect, arcing of the current occurs, and this burns away the electrode. This condition can often be recognized when the joint is still some distance above the roof, by the appearance of red hot spots on the electrode, when it is still black below the joint. After the electrode has been fed down far enough for the joint to be below the roof, the joint begins to burn away on the outside also, the opening forming a point where oxidation progresses more rapidly than on the rest of the electrode. The joint then becomes much weakened, and finally breaks under the weight of the electrode, or from bending in lowering the electrode upon the irregular heap of cold stock. Thus a considerable piece of each electrode is frequently lost.

This joint problem limits the length of the electrode, on account of the greater weight of the longer pieces, and also its diameter, because if too small an electrode is used, the walls around the plug will be thin, and after only a small amount of joint wear ("wasping") has occurred, the electrode will break. This remark is more applicable to graphite than to carbon electrodes, and is somewhat offset by the fact that the threads of graphite electrodes are usually made to a much better fit.

The openings in the roof through which the electrodes are introduced are provided with hollow bronze or cast-iron rings, cooled by water circulated through them; generally the water passes first through the bronze electrode holders, then through the roof coolers. The latter are set upon the roof and connected to the water pipes by joints that can be readily broken, so that a cracked cooler can be replaced with the least possible delay. The coolers should fit the electrodes as closely as possible, without causing the latter to stick. If the fit is too close, an electrode that is a little out of round will often stick; it can be chipped to pass through the holder, but the chipped place will burn away more rapidly than the rest of the surface. Too large an opening makes a leaky furnace and causes the electrodes to burn away to a serious extent above the roof. Of late several devices have been invented to reduce this space to a minimum

without making the opening in the coolers themselves so small as to result in sticking of the electrodes.

The doors of the electric furnace must be so designed that all openings into the hearth can be tightly closed. In fuel-heated furnaces this is seldom necessary because there is a certain amount of pressure within the furnace that causes the flame to come out of the cracks; but in the electric furnace the hot gases tend to flow out between electrode and roof cooler, creating a draught that draws air in through any opening around the doors. If this inflow of air is considerable, it burns away the electrodes rapidly, and what is worse, makes it impossible to maintain neutral or reducing conditions in the slag.

Some furnaces have been built in which the door consists chiefly of a tapered block or plug made of one piece of brick, which fits the door jamb closely, in the same way that the breech-plug of a big gun fits into its seat. This design is suitable only for very small furnaces, and is never really satisfactory, because the joint soon gets loose from the burning away of the brick. The best practice appears to be to have the door arranged to slide freely in a frame attached to the outside of the furnace, and so designed that the door can be pushed back tightly against the brick jambs and arch by means of a wedge. The frame should extend out at the bottom far enough to hold a few shovels-full of lime or sand, which will seal the opening at the bottom. The spout of the furnace serves the same purpose for the door at the front. If the doors are made too large, it is virtually impossible to seal them tightly enough.

The acid furnace is usually lined throughout with silica brick, and the bottom is of silica sand sintered to the brick. The bottom of basic furnaces is lined with magnesite or metalkase brick, and the same material is used for the side walls up to a few inches above the highest point to which the banks are to be built. Above that point, the walls are usually of silica brick, though sometimes a patch on the back wall is made of magnesite or metalkase, because in the common triangular arangement of a three electrode furnace, one electrode is quite near the back wall and the wear on the lining at this point is more severe than anywhere else.

Sometimes one course of brick next to the shell is of clay brick, but this practice is not to be recommended, as the clay brick so reduces the radiation through the walls as to shorten greatly the life of the silica brick. The door jambs, except at the very bottom, are of silica brick, and of course the door arches and roof are of the same material. Various special refractories have been tried for electric furnace linings with varying degrees of success, but none of them can be regarded as standard practice as yet.

Lay-out.—The arrangement of electric furnace melting shops is usually quite a simple matter, as the furnaces are comparatively small for the tonnage produced, and occupy very little space. In most shops they are set at floor level, to avoid the necessity of hoisting the scrap, alloys, etc., to a charging floor; in this case, the tipping mechanism is placed in a pit below the ground level, and a pit is needed for the ladle. The transformers are commonly placed upon an elevated platform in a room outside the melting shop walls, so that the leads to the electrodes can be brought down in a long loop to the furnace. With the bins to hold alloys, lime, etc., located conveniently and proper space provided for lining and drying out ladles and for rebuilding roofs, the lay-out is complete.

Except for very small furnaces, the yard for storing scrap and the bins where alloys, etc., are kept in bulk should be so arranged that scrap charges can be loaded on small cars and brought to the furnaces; one often sees shops where wheelbarrows are used for this purpose, but the extra expense for wheeling is considerable.

The furnace, of course, should be spanned by a crane to handle the ladles, roofs and electrodes; in most shops the furnace is located near one wall of the main foundry building and the traveling cranes that serve the moulding floors take care of the furnace also.

Though not exactly a question of lay-out, the choice of a proper size of furnace for the work to be made is of great importance. In general, electric furnaces in the foundry find their best field in the production of light castings; it is well, therefore, not to install too large a furnace, on account of the difficulty of pouring a large amount of metal into small work. If there is enough business in sight, for instance, to require one 6-ton furnace, or two 3-ton furnaces, the latter will be the better choice. For the occasional castings requiring more than 3 tons of metal, both furnaces can be got ready to pour at once, but most of the work will be comparatively light, and can be poured better with a 3-ton than with a 6-ton heat. Then, again, when business is poor one furnace can be shut down. These considerations in most cases outweigh the slightly lower operating costs of the larger furnace.

Setting the Bottom.—When electric furnaces were first introduced, the bottoms were tamped in place while the furnace was cold, coal tar or some other binder being mixed with the magnesite or

sand to hold it in place. The furnace was then heated, and an attempt made to set the bottom. Only the top layers could be properly sintered, and a bottom so made was most unsatisfactory. To-day, the bottom material is sintered in place an inch or so at a time, each layer being heated to partial fusiont before the next is applied. This can be done by putting in a layer of coke, lowering the electrodes upon it, and turning on the current, but this method is unhandy, as the hot coke has to be scraped out after each heating. The accepted practice to-day is to place three pieces of electrode on the bottom with their ends together at the middle of the hearth, as children place broken fire crackers to make a "sizzer." A tight joint should be made where the three pieces meet, by cutting their ends to a proper angle, and cementing in a joint plug with joint compound. One electrode is lowered against each of the pieces, and the current turned on. The resistance at the contacts and in the short pieces of electrode heats the furnace. The brickwork should be dried out for 6 or 8 hours; when it is thoroughly dry the temperature can be brought up to a sintering heat and the first layer of bottom placed. When this has been sintered tightly to the brickwork, the next layer is shoveled in, and the process continued until the bottom is completed. Each layer is heated until it sticks together; a bar is used to make sure the material is "set."

The pieces of electrode used for resistors are pried up with bars as the bottom is built up, as when the furnace is hot, sufficiently good contact is made if they touch each other at their ends, and the tight joint is no longer necessary. The operation of setting a bottom requires from 24 to 36 hours or longer, depending upon the size of the furnace, and the whole bottom should be put in without interruption; the best demonstrators in the employ of the reliable furnace builders usually supervise the entire operation themselves, as none realize better than they how important it is to have the bottom as nearly perfect as possible. When the bottom is completed, a "wash heat" of scrap and pig iron may be melted to help set the material; the good demonstrators, however, will set a bottom well enough so that the first heat may be used for regular production, and no "wash heat" need be used.

The material for the bottom of the acid furnace is silica sand such as is used for steel molding, containing 95 per cent. silica or over. For the basic furnace, calcined magnesite is almost always used. The low-grade magnesites containing considerable proportions of

lime, and the various calcined dolomites are not satisfactory for making a bottom, or for filling up any large holes that may have to be repaired. They are extensively used, however, to patch the slag line after each heat.

With the magnesite should be mixed about 10 per cent. of ground basic open-hearth slag to assist in sintering the bottom. The imported magnesite and most of the prepared domestic material sinters at so high a temperature that the slag is required to make it set rapidly enough.

The Basic Process.—Usually, cold scrap such as rail butts, punchings, turnings, etc., of such a composition that the phosphorus can be brought to .03 or .04 per cent. with one "slagging off," is used for the charge of the basic furnace. It does not pay to melt scrap with phosphorus much over .1 per cent., as the extra cost of refining when more than one slag has to be scraped from the furnace more than balances the saving in the cost of the raw material. A greater proportion of light material can be successfully melted in the electric furnace than in any other process, but it is well to avoid going to extremes in the use of very light scrap, especially turnings. If it is proposed to melt a considerable proportion of turnings, only thick heavy ones should be used, and they should be clean and free from rust. In general, at least one-third of the charge should be of fairly heavy material.

In charging the furnace, care should be taken to compact the mass of scrap well together, so that the current will pass freely through the metal. If the charge is thrown in higglety-pigglety like a heap of jack-straws the furnace will be too full, and there will be a good deal of trouble experienced in starting up. If the top of the heap is very irregular, moreover, the electrodes may break from side-thrust in lowering them to make contact. By charging part of the heavy scrap, then some of the light material, then more big pieces, and so on, a compact mass can be built up. It is sometimes also a good scheme to leave a small portion of the previous heat in the furnace to cement the bottom of the charge together.

Part of the lime is usually charged with the scrap, generally on top of the heap, at the electrodes. The electrodes are lowered until they nearly touch the charge, the current turned on and the automatic controllers thrown in; the electrodes then quickly feed down till the current starts. If there is any trouble getting the arcs started, some heavy turnings can be shoveled under the electrodes, or a bar held under their ends to assist in making good contact.

Sometimes a small piece of old electrode is thrown under an electrode if it gives considerable trouble, but if charging has been properly done this should not be necessary. For perhaps 10 or 15 minutes, the melter has to watch the instrument board, and raise or lower the electrodes to renew the arcs or to cut down the current at one arc or another when the amount of current flowing gets too high. During the melting down, as much current is used as the furnace can take, in order to melt the steel in the shortest possible time.

The arcs soon become steady, and the electrodes "bore down" through the charge as the metal melts under them, forming pools of liquid steel. As soon as these pools are well started, a few shovelsfull of lime should be thrown in at each electrode, to protect the metal from oxidation. The electrodes feed down until they nearly reach the bottom of the furnace, then as the bath of liquid metal begins to fill the furnace, they gradually come up again as the automatic controllers keep the arcs at a constant length.

When the charge is a little more than half melted, it is generally necessary to open the doors and poke the scrap off the banks with bars. This operation may have to be repeated before all the metal has been gotten into the bath; and when small graphite electrodes are used, it may be necessary to break up with bars a mass of scrap lying between the electrodes and "bridging over" the bath. It is on account of this necessity for barring in scrap that furnaces of over 6 or 8 tons capacity are seldom used for melting cold charges, the labor of poking in the scrap in a larger furnace being too severe and the melting down period too long.

The bath when melted will be covered with a slag composed of the lime added, (about 40 lb. per ton of steel), the iron oxide from the rust on the scrap and from oxidation during melting, and some manganese oxide and silica from the latter source. If the scrap is very clean and free from rust, a little iron ore or hammer scale must be used to provide sufficient iron oxide to oxidize the phosphorus, but this is seldom required. Occasionally a small amount of fluorspar is needed to thin up the slag and assist it to dissolve the last of the lime. If much sand adheres to the foundry scrap it will make the slag too acid, and extra lime will be needed to take care of this silica, because unless the slag is very basic it cannot retain all the phosphorus in solution as phosphate of lime, which is formed by the union of CaO of the slag with P_2O_5 resulting from the oxidation of the phosphorus.

When the metal and slag are entirely melted, a test piece should be

taken to determine the carbon content of the bath, and if very pure steel is desired, the phosphorus also. In ordinary practice, using scrap of about .08 or .1 per cent. phosphorus, the latter will be reduced to .02 or .03 per cent. by the time the slag is all melted and the slagging off may follow immediately.

Should the fracture of the test piece indicate that the carbon is too high, iron ore or scale is thrown into the furnace, generally before slagging off, and a second sample taken when the ore has had time to work on the bath and the fracture looks about right in carbon. When the second or de-sulphurizing slag has been formed, additions of iron oxide are, of course, undesirable; the carbon, therefore, should be brought a little below the content desired in the finished steel before de-sulphurizing starts.

Current is taken off, the electrodes raised out of the way and the slag raked out of the furnace through the spout door with rabbles; bars may be introduced through the side doors also to assist in skimming the slag toward the front of the furnace. The bath should be scraped quite clean, as the phosphorus in any slag left in the furnace will be reduced and re-enter the steel as soon as de-sulphurizing is begun.

The second slag is made up of lime, to which is added a very little silica sand and enough fluorspar to form a liquid slag in a reasonable length of time; about 40 lb. of lime, 20 lb. of spar and 5 lb. of sand per ton of steel, are commonly used. As soon as the second slag is melted, the current should be considerably reduced, and as little power carried as will heat the steel to the desired degree. When the metal is nearly ready, it can be brought rapidly to temperature by increasing the current for 5 or 10 minutes.

The slag at first will be somewhat thin and black. At intervals, fine coke dust, preferably of petroleum coke, is spread over the surface of the bath with shovels, to reduce any oxides of iron and manganese in the slag. As this reaction progresses, the slag, which at first may be brown or green from the oxides, becomes cream colored or even white on a fresh fracture. When the bar or test spoon is withdrawn from the furnace it should be cooled in water and the slag adhering to it inspected. This slag, once it is properly deoxidized, will crumble to a white or cream-colored powder in a few minutes, and the steam rising from it will often have an odor of acetylene from the calcium carbide formed by the combination of the lime and the carbon of the coke dust. Occasionally when de-oxidation is carried very far, so much calcium carbide is formed

in the slag that it becomes dark or even black, crumbles to a blackish powder, and gives off acetylene.

When it is properly deoxidized, the slag in the furnace will foam up as the coke dust is spread over it, and the arcs will become nearly or quite buried in the foamy slag. This is an excellent indication that complete deoxidation is being attained. Generally, half an hour after the second slag is made is a sufficient length of time to deoxidize the metal thoroughly. The oxides of iron and manganese, of course, are removed by reacting with the carbon of the coke dust, forming metallic manganese and iron, and CO which escapes as gas. The sulphur is converted to calcium sulphide, and dissolved in the slag; some appears to escape as SO₂ by oxidation of the CaS.

Calcium carbide is formed in the slag after the metallic oxides are reduced, through the reduction of CaO to Ca by the carbon, and union of this Ca with some of the coke. It was formerly supposed that this CaC₂ rather than the lime of the slag, was largely responsible for the elimination of the sulphur, but opinion of late years inclines to the view that the calcium carbide is an indicator of complete deoxidation and desulphurization, rather than an active desulphurizer.

As deoxidation progresses, samples should be taken at intervals to judge the temperature of the metal. The spoon is well heated up, then coated with slag, then dipped deep into the bath and withdrawn. The sample should be taken from near the bottom of the bath, because the metal there is naturally the coldest. The slag is scraped from the metal in the spoon, and the number of seconds before the steel begins to freeze on the surface is noted. For ingot work, 10 seconds indicates that the steel is hot enough; for very light castings, as high as 60 seconds is often necessary.

Some furnace operators add ferromanganese and ferrosilicon to give nearly the desired amounts of those elements, after the first slag has been skimmed off, and make only small final additions before tapping. Others charge about half the alloy after slagging off, the rest 5 or 10 minutes before pouring. The alloys may be thrown in, together with some pig iron if any is needed to bring the carbon to the desired figure, 5 or 10 minutes before pouring the heat.

In order to secure the minimum loss of manganese and silicon in the recarburizers, it is necessary to have the slag quite thoroughly deoxidized before making the final additions. If the slag cannot be brought to a deoxidized state, on account of excessive air leaks around the doors, improper lime containing too much alumina or magnesia, too great contamination of the slag with magnesia from a badly repaired bottom, or what not, these losses, especially of manganese will be very great, and what is worse, they will vary so widely from heat to heat that control of analysis will be impossible. A little coke dust should be spread on the slag immediately after adding the alloys; as soon as they are melted and the bath hot enough, the steel is poured into the ladle and taken to the pouring floor

When no skimming device is used to keep back the slag in pouring, so that steel and slag run into the ladle together, care must be taken to see that the last coke dust spread over the slag is thoroughly "burned off." If much fresh coke dust is floating on the slag when the heat is poured, it will be churned up with the steel and sometimes raise the carbon to a surprising degree. A "carbide slag" will often do the same thing. This is true also, to a less extent, of coke on the slag in acid practice; less because the acid slag is so viscous that it does not flow out so freely and mix as thoroughly with the steel.

In some cases ferrosilicon is used in addition to coke dust to deoxidize the slag and the bath. For certain alloy steels its use is justified, but a skilled melter operating a good furnace should be able to deoxidize the metal thoroughly with coke alone, and it is even stated that the free use of ferrosilicon in a basic furnace is a sign of a poor melter.

As soon as the steel has been poured into the ladle, the furnace should be drained quite clean, and the metal rabbled out of any holes in the bottom. These holes and the banks at the slag line are then patched with dolomite or magnesite, and the furnace re-charged. Failure properly to splash out a hole in the bottom before patching is very dangerous, as it often results in the development of a large hole that either grows until a heat breaks out, or necessitates a shut down to re-build that part of the bottom.

Electrodes sometimes break off and stick in the half-melted charge of scrap. As much of the electrode as possible should be removed from the furnace, as the carbon of what remains raises the carbon content of the bath so high that much time is lost and iron ore used in reducing the carbon to the desired point. A small piece of electrode falling into the furnace after the steel is melted is not a serious matter, as it is usually easy to get it out promptly; but if the heat is nearly ready, such an accident sometimes results in quite a loss of time because the carbon has to be brought down with iron ore, and the slag then deoxidized over again. The experienced operator

watches his electrodes closely, knocks off pieces that appear likely to fall, and removes an electrode that threatens to break near the joint. With careful supervision, trouble of this sort is reduced to a minimum.

When very low phosphorus is desired in the steel, or when the phosphorus in the scrap is high, it may be necessary to make and remove two oxidizing slags to secure the analysis desired. Unless there is a very good reason for securing extremely low phosphorus in the steel, however, the extra expense should be avoided; and if two slags are being raked off to refine impure scrap, it may be found to pay to buy a little better raw material and save the time, current, etc., lost in this slower process.

When liquid metal is used as the charge of the furnace, the oxidizing slag for the removal of phosphorus is formed by shoveling in the lime, ore or scale, etc., as soon as the metal is charged; the process from that point on is the same as though cold scrap had been melted. Of course, if dephosphorized metal from a basic open-hearth furnace is to be refined in an electric furnace, only the deoxidizing slag is needed.

One Slag Operation.—When a supply of cheap low-phosphorus scrap is available, it may be cheaper to melt this material than to refine still cheaper scrap containing higher phosphorus. This is the established procedure in acid practice, but strangely enough few basic furnaces are so operated. With scrap containing less than .04 per cent. phosphorus, the basis process can be greatly speeded up by making the first slag the deoxidizing and de-sulphurizing one; in operating by this method, the first slag is treated with coke dust until it is properly deoxidized, the alloys added, and the metal poured. The sulphur is thus brought to very low percentages, the steel thoroughly deoxidized, and a metal considerably better than open-hearth steel is produced. If the opinion of leading authorities is correct that basic electric steel is per se better deoxidized and of better quality than acid, this is the logical process to use, instead of changing the furnace over to an acid lining.

The Acid Process.—The bottom of the acid electric furnace consists of sintered silica sand; hence the oxide of iron from rust and scale on the scrap and from oxidation of iron in melting down, and the oxide of manganese from the manganese oxidized, which are strong bases, dissolve enough silica from the bottom to form a slag consisting essentially of a silicate of iron and manganese, containing usually about 50 to 60 per cent. silica and 40 to 50 per cent.

FeO and MnO. As has been explained in considering the openhearth process, such a slag will not retain phosphorus in combination, because the silica has a greater affinity for bases than has the P_2O_5 , and the latter, left uncombined, would be reduced again as fast as it was formed. Similarly, no sulphur can be eliminated from the charge. The raw materials for the process, accordingly, have to be a little lower in phosphorus and sulphur than is desired in the finished steel, as there is theoretically a slight gain in these elements due to the loss of iron in melting down.

Unlike the basic process, therefore, in which the composition and volume of the slag are under the control of the operator, the acid process is one in which the amount of iron oxide in the form of scale and rust on the scrap, and from oxidation in melting due to air leaks around the doors, controls not only the composition of the slag but its volume as well. As the amount of iron oxide varies widely from time to time, depending upon the size of the scrap charged and its condition, the volume of the slag varies widely. In basic operation, it is common practice to charge a heat one night and melt it the next day; indeed sometimes current is turned on the night before long enough for the electrodes to "bore down" through the charge. When scrap is left in the acid furnace over night, the amount of scale it will form will depend upon the condition of the furnace and the care used in mudding up the doors, introducing a further variable.

The slag, therefore, when the heat is first melted, is of a highly oxidizing nature, and to produce a deoxidized steel it is necessary to alter its composition so that it shall be as little oxidizing as possible. Except in occasional instances, it is not, of course, necessary to remove this slag, as in the basic process; indeed the chief reason for the existence of the acid process is that it is a one-slag operation, and hence can be carried out more rapidly than the two-slag basic process.

It is not feasible to reduce with coke dust the iron oxide of the slag as it exists after melting, because a silicate of iron with silica in excess is difficult to dissociate, and the iron oxide being the prevailing base is far more stable than in the basic slag, where the chief base is lime. In the basic slag, as it were, the iron oxide is but a side dish and if it is taken away there remains the lime to satisfy the appetite of the silica; while in the acid furnace the iron oxide is all the silica has to satisfy it. There is a chemical law which says that solutions and liquid slags have a great affinity for any substances that if

dissolved will reduce their melting point; as a natural consequence, to remove such a constituent of a slag is very difficult. Should the iron oxide be removed, it would leave a large part of the silica uncombined, raising the melting point many hundred degrees. Further, if coke dust is spread over the acid slag immediately after melting down, silicon will be reduced from the silica of the slag by the carbon of the coke, especially at the arcs where the temperature is high enough to reverse the affinities of carbon and silicon for oxygen, and silicon will enter the steel in large percentages.

Ferrosilicon also will not tend to reduce the iron oxide content of the acid slag beyond a certain point, because the reaction between the silicon and the iron oxide produces iron and silica, and the latter must enter the slag; and when the silica content of the slag reaches a point where further increases will raise its melting point, the reaction ${}_2F_2O + Si = SiO_2 + {}_2Fe$ no longer takes place, but the ferrosilicon melts as such and the silicon enters the steel.

It is perhaps not an over-statement to say that in some shops using the poorer types of acid furnace, the operator "passes the buck" to the furnace itself; he melts his charge, keeps heat on the furnace till enough silica has been absorbed from the bottom to bring the slag to a stable condition, adds some ferrosilicon and ferromanganese to bring the composition about right, then pours his steel into the ladle. Indeed, the author was credibly informed that one such operator, when asked how he judged that the heat was ready to pour replied "Why the roof is beginning to run; that's my usual signal!"

Other operators, a little more skilful, make small additions of iron ore if the silica in the slag appears to be too high, or "work" the slag and bath with coke dust and ferrosilicon if bases are in considerable excess. When they have brought their slag to a fair condition, and have added enough silicon to the steel so that the tests solidify quietly in the mould and show a fracture free from blowholes, they add the ferromanganese and pour their heat.

Before an effort is made to reduce the iron oxide content of the slag, the carbon of the bath should be brought slightly below that desired in the finished steel, because to add iron ore to lower the carbon content, after the iron oxide content of the slag has been reduced, undoes the work just completed and produces a thin.

¹ When the slag happens to melt down of such a composition that silica is in excess, the composition adjusts itself through the reduction of silicon from the excess silica of the slag by the reducing action of carbon.

oxidizing slag again. According to good authority, the proper way to reduce the iron oxide from the slag is to substitute for it other bases, at the same time making additions of coke dust. Lime up to 20 or 30 per cent. of the slag volume should be used, and if manganese ore low in iron is available at not too high a price, it may be added until the slag contains 10 to 15 per cent. MnO. When enough lime has been added and the iron oxide content sufficiently reduced, the slag becomes bluish or greenish on the fresh fracture, and of a glassy appearance, quite free from sponginess or porosity; it is in fact almost a true glass. When cooled on the bar or the handle of the test spoon, its surface turns brown or black, the latter if there is still too much iron oxide in it, brown if the iron oxide is low enough.

When the slag has been brought to this composition it is as nearly non-oxidizing as it can be made, and the losses of the manganese and silicon alloys used as re-carburizers will be a minimum. A typical analysis of such a slag² is SiO_2 58.7 per cent., CaO 21.25 per cent., MnO 12.01 per cent., FeO 3.10 per cent. ,MgO 1.15 per cent., Al₂O₃ 3.35 per cent.

In the admirable paper presented before the American Electrochemical Society, Mr. Lindemuth explains in detail the tendency of the acid slag to assume a definite proportion of base to acid, and gives examples of slags of different analyses which are of the same degree of basicity, figuring all the bases present to their equivalent proportions in one of the bases. He demonstrates also the fact that the slag in the acid process is never wholly or even nearly freed from oxide of iron, and hence remains at least slightly oxidizing throughout the process; as a result, in Mr. Lindemuth's opinion "it is very questionable if under the most skilful supervision, acid electric steel can be made of a quality equal to that produced by the basic process."

The additions of ferromanganese, ferrosilicon, and pig if the carbon is too low to be brought to the required figure by the carbon of the ferro alloys alone, are made when the slag has reached correct composition, coke dust being spread over the slag after the additions are charged to minimize the losses. When the additions are melted and the steel hot enough, the heat is poured into the ladle, as much as possible of the slag remaining in the furnace is hoed out, any

¹Lindemuth, The position of the electric furnace in iron and steel metallurgy: *Proc. Am. Electrochem. Soc.*, vol. xxxvii (1920), p. 299.

² Lindemuth, "The acid electric furnace process: Trans. Am. Foundrymen's Asso., vol. xxviii (1919), p. 232.

holes in the bottom splashed out and repaired with sand, the banks patched with the same material and the next heat charged.

The acid process has been adopted in foundries for three reasons. The first is that the tonnage produced is greater than that of the basic process as usually operated, so that in many cases the acid steel is the cheaper. As we have seen, the basic furnace can re-melt low phosphorus scrap as rapidly as the acid furnace, so that this reason alone will not justify the use of the acid process. The second reason appears to be a feeling that just as acid open-hearth steel is better than basic, so acid electric steel should be the better. As we have seen, competent opinion does not endorse this view.

The third reason for the adoption of the acid process is that as the slag is very viscous and of high melting point, it can be readily kept out of the hand ladles and the moulds, while the more fluid basic slag gives a good deal of trouble in pouring. The acid slag freezes over soon after the heat is poured into the ladle, and, especially if a good skimming device is used in lip pouring, it is easily kept out of the moulds or the hand shanks. With skimming lips on the latter, small castings can be poured with very little trouble from slag.

This difficulty can be overcome by using a bottom-poured ladle, but for small work lip pouring is undoubtedly better than the use of a nozzle. The "coffee-pot" ladle was designed to allow the steel to be poured "over the lip" without skimming off the slag, so that the latter can be retained in the ladle to assist in keeping the steel hot; but the coffee pot ladle is not always satisfactory. For foundries so equipped and laid out, therefore, that the pouring of small castings cannot be handled very rapidly, the acid process undoubtedly assists in the pouring of clean castings.

Where the ladle can be skimmed clean and poured into castings in a comparatively short time, so that the steel does not grow too cold from exposure to the air to pour freely, fluid slag should not be an insurmountable obstacle to the use of the basic process and lippoured ladle. Even when small ladles have to be used to pour all or a large part of the heat, in a well laid out shop the steel can be taken care of fast enough to allow the ladle to be skimmed clean without danger of trouble from cold steel. With a pouring gang familiar with handling fluid slag, no trouble has been experienced in the shop with which the author is connected in pouring basic electric steel into castings of all weights from a few ounces up to several tons. In this shop, too, the habitual use of a magnesite lining for the ladles, which will be referred to later, undoubtedly

lessens the difficulties which most foundries encounter in handling basic electric furnace metal in lip-poured ladles.

The Raw Materials.—For the acid process, of course, the scrap must be "Low phosphorus," containing less than .04 per cent. phosphorus and sulphur. A great variety of light and medium sized scrap can be used, and heavy turnings up to 30 per cent. of the weight of the heat may be melted. Very rusty material should be avoided, for reasons given in the discussion of melting practice.

It is not usual to melt metal containing much over .r per cent. phosphorus and .o5 to .o8 per cent. sulphur in the basic electric furnace, as the extra time and current consumed in the refining more than balance the saving in the cost of the raw material, especially if scrap is bought so high in phosphorus as to require two dephosphorizing slags. Rust is not so undesirable as in acid work, but very rusty scrap, especially light material and turnings, should not be used freely.

The lime for the basic process must be very pure, containing as little alumina and magnesia as possible, and preferably at least of per cent. CaO. Limestone is seldom used in place of burned lime, partly because it has to be calcined in the furnace, which delays operations and consumes extra power, and partly because it is often contaminated with excessive dirt. A great deal of the burned lime obtainable is quite unsuited for use in the electric furnace, especially that containing much magnesia. Lime should be bought to specification from a concern able to furnish a uniform material, and one free from contamination with coal ashes. If the lime contains too much magnesia and alumina, or both, it will be impossible to secure a properly deoxidized slag, control of analysis will be difficult and inferior steel will be produced. To a smaller extent, the same is true of the fluorspar used to thin the de-sulphurizing slag; if it contains too much alumina or magnesia, trouble with the slag is apt to result. A spar containing 85 per cent. CaFl is considered excellent. A good analysis is as follows:

 $SiO_2 - 2.04$ per cent. $Al_2O_3 + Fe_2O_3 - 11.64$ per cent. $CaCO_3 - 1.00$ per cent. CaFl - 85.32 per cent.

The magnesite used for the bottoms should not contain too much lime, because when the content of CaO is too high the material slakes like burned lime on contact with the air. A bottom made of impure magnesite will slake quite badly if the furnace is cooled down for several days. For patching, however, it is not necessary

to use the high-grade magnesite, as the more impure material or even calcined dolomites will serve. Typical analyses of magnesites and dolomites are as follows:

C. Doz ostron

| CALCINED DOLOR | MITE | | |
|--|-------|-------|--------------|
| | | | Per cent. |
| SiO_2 | | | . 12 |
| $Al_2O_3\text{-}Fe_2O_3.\dots\dots\dots\dots\dots$ | | | . 9 |
| CaO | | | |
| MgO | | | . 32 |
| CALCINED MAGNI | ESITE | | |
| | (1) | (2) | (3) |
| | Per | Per | Per |
| | cent. | cent. | cent. |
| SiO ₂ | 7.40 | 8.40 | 6.30 |

2.50

2.60

5.40

4.30

79.35

.40

1.40

13.20

71.30

Number 3 is rather too high in CaO.

Al₂O₃.....

Fe₂O₃..... 8.60

MgO..... 79.2

2.80

Ordinary coke dust can be used for deoxidizing the slag, but petroleum coke is preferable, as it is almost pure carbon and contains very little sulphur or phosphorus and practically no ash to contaminate the steel and slag. It is a by-product from the refining of petroleum and can be obtained from several sources.

Pig iron is never used as part of the charge of an electric furnace, first because it usually costs more than scrap, and second because the removal of much carbon from the bath takes too much time and current. For the same reason, to handle cupola metal in an electric furnace would not pay; we have discussed in an earlier chapter the possibilities of "triplexing" in a properly designed plant, using the cupola, Bessemer converter and electric furnace. Such a process, however, cannot as a rule be successfully introduced in an existing plant, as in the majority of cases the lay-out and pouring methods could not be arranged to suit the process.

Open-hearth Electric and Bessemer Electric.—In an effort to save power in melting, a combined electric furnace and Bessemer converter has been proposed, but as far as the author is aware none has ever been built. It would, of course, be almost out of the question to

design such a piece of apparatus that would operate successfully.

The combined open-hearth furnace and electric furnace appears more attractive, but some years ago a careful trial of such a furnace proved a failure; it was neither a good open-hearth furnace nor a good electric furnace, and moreover it was cumbrous to handle and the closing off of the gas uptakes and the placing of the electrodes required too much time.

To take advantage of the lower melting costs of the open-hearth furnace or the cupola and converter, using the electric furnace only for the final refining, a duplex or triplex plant is the only possible solution. Large shops can economically refine open-hearth metal in an electric furnace. Basic open-hearth metal already low in phosphorus and sulphur may be finished and further refined in either a basic or an acid electric furnace; or acid open-hearth steel may be treated in a basic electric furnace. The triplex process, using cupola, Bessemer converter and electric furnaces proved a failure in at least one shop that attempted it, but the success of the process for the production of malleable iron in a well-designed and well-operated shop shows that it can be successfully operated for the production of steel. In many cases, however, it will not result in a saving, as the higher cost of the pig iron required will more than offset the cheaper melting in the cupola.

Alloy Steels.—In the manufacture of alloy steels in the electric furnace, very few such difficulties are encountered as are common in the other processes, because almost any metal or ferro alloy can be added to the bath, especially in the basic furnace, with but slight loss of the alloy. Thus nickel, ferrochrome, ferrovanadium, ferrotungsten, etc., are added with the recarburizers and practically their entire metallic content enters the steel. Similarly, scrap containing these alloys can be re-melted in any amount up to the entire weight of the charge, and the alloys recovered. A certain amount of the more readily oxidized metals will be converted to oxides during melting down and will enter the slag. These will be lost in "slagging off" the basic furnace, if high phosphorus metal is being refined; in the acid process, only a part of the metals so entering the slag can be reduced and returned to the bath. will be best, therefore, when chrome- or vanadium-steel scrap is to be re-melted as part of the charge in a basic furnace, to use lowphosphorus stock for the balance of the heat, so that slagging off will not be necessary.

Manganese steel scrap, containing usually about 1.25 per cent.

carbon and 11 to 13 per cent. manganese, can be successfully melted only in the basic furnace because the large amount of manganese oxide formed in melting this steel would cut deeply into an acid bottom and produce great volumes of slag. To reduce the manganese from such a slag would be almost impossible, and the loss of manganese would be very costly.

Another difficulty encountered in the re-melting of manganese steel scrap is that if an electrode breaks off in the scrap or falls into the bath, the carbon absorbed by the steel cannot be removed, without first eliminating the manganese, because oxygen has a greater affinity for manganese than for carbon. If iron ore is added, therefore, all the manganese will be oxidized and slagged before the carbon can be reduced, and the subsequent reduction of such a quantity of manganese from the slag with coke dust would be practically impossible. Theoretically, the carbon can be reduced without loss of manganese, by adding manganese ore to the slag; there should, in fact, be a gain in manganese, because the carbon is reduced by the reaction C + MnO = CO + Mn. The carbon of a bath of 80 per cent. ferromanganese has been reduced by this process in an electric furnace from over 6 per cent, to less than 1 per cent. For some reason not readily apparent, however, manganese ore will not react with the carbon to the exclusion of the manganese in a bath of 12 per cent. manganese steel, and no one to the author's knowledge, has succeeded in the attempt to make real use of this principle in the re-melting of this scrap. It is necessary, therefore, to be very careful that only good electrodes are used, and if a piece does fall into the bath, no time must be lost in removing it. By careful watching of the electrodes, it is possible to re-melt the scrap with the loss of but a very small percentage of heats on account of high carbon. Again, it is essential that the furnace be as air-tight as possible, in order to minimize the oxidation of manganese in melting down. Some is volatilized and lost in any case, so that it is essential to reduce the amount oxidized and slagged to the lowest possible point.

It is more difficult to start the current in melting manganese steel scrap than in the case of ordinary steel charges, because manganese steel is a very poor conductor of electricity. Particular care, therefore, is necessary in charging; in some cases, too, a small part of the previous heat is left in the bottom of the furnace to stick to the new charge and assist in starting up the arcs.

No slagging off is required in re-melting manganese steel scrap;

in fact, to slag off would be to lose a considerable proportion of the manganese, as the manganese content of the slag when the heat is first melted is quite high. A preliminary test for carbon and manganese should be taken as soon as the bath is completely melted. The slag should then be deoxidized as completely as possible, additions made to make up for the small percentage of manganese lost in melting down, and the heat poured as soon as it is hot enough to run the castings.

CHAPTER VII

SUMMARY—SPECIAL DEOXIDIZERS—LADLES

As a convenient reference, the changes that take place in the content of metalloids in the several steel-making processes are summarized as follows:

| | | Element | | | | | | | | | |
|--|------------------|--------------|--------------|--------------|--------------|--------------|------------------|--------------|--------|------------------|--------------|
| Process | | С | Si | Mn | S | P | Ni | Cr | W | Mo | Va |
| Cruci- ble | graphite | gain | (gain) | loss | (gain) | (gain) | | (loss) | (loss) | (loss) | loss |
| | clay | loss | (gain) | loss | (gain) | (gain) | | (loss) | (loss) | (loss) | loss |
| Bessem | er (acid). | loss | loss | loss | gain | gain | (loss) | loss 3 | loss | (loss) | loss |
| Open hear | acid th basic | loss loss | loss loss | loss loss | gain loss | gain loss | (loss) (loss) | loss loss | loss | (loss) (loss) | loss loss |
| Electric Melt (Basic bottom) Melt and refine | Melt | (loss) | (loss) | (loss) | (loss) | (loss) | | | | | |
| | | loss | loss | 1css | loss | loss | | loss | loss | (loss) | loss |
| Electric (acid bottom) | | (loss) | gain | (loss) | (gain) | (gain) | | (loss) | (loss) | (loss) | (loss) |

(gain) or (loss) indicates slight gain or loss.

gain or loss indicates practically complete gain or loss.

..... indicates neither gain nor loss.

Special Deoxidizers.—In addition to the regular recarburizers, it is frequently necessary, especially in making very low carbon steel, to use other materials to more thoroughly deoxidize the steel and make it fluid and clean running. Aluminum has been used for a number of years for this purpose and has greatly aided the steel foundryman in pouring sound castings. It is a more efficient deoxidizer than silicon or manganese, and the proportion of the oxides reduced is much increased by its use. There are, however, several objections to it, the most important of which is that the product of the reaction,

$$_3$$
FeO + $_2$ Al = Al $_2$ O $_3$ + $_3$ Fe

is alumina (corundum), an almost absolutely infusible mineral, which does not coalesce and float out of the steel but remains in

finely divided form in the metal, and renders it somewhat weak. In rolled steel, especially plates, the effect of this alumina is well known to be a considerable reduction of the strength of the steel when tested in a direction transverse to the direction of rolling.

The first of the special alloys to be used as final deoxidizer or "wash" was "S-A-M metal," which contains generally about 10 per cent. silicon, 5 per cent. aluminum and 10 per cent. manganese. The products of the reaction of this alloy with FeO are SiO₂, Al₂O₃ and MnO, which unite to form a double silicate of manganese and aluminum, of comparatively low melting point. Such a silicate will coalesce into globules of sufficient size to float out of the steel to a considerable extent. Moreover, as Mr. Hibbard¹ has shown in his suggestive article on this subject, globules of slag of considerable size are less harmful to the steel than very minute solid particles scattered broadcast throughout the metal.

Most of the other "washes" now on the market are designed to give reaction products of great fluidity. For instance, the alloy of calcium and silicon produces CaO and SiO₂, which are counted on to unite with the MnO resulting from the reduction of FeO by manganese, to form a fusible silicate of CaO and MnO.

A somewhat different object is aimed at in the use of ferrotitanium as a final addition to steel. Titanium has the property of burning in an atmosphere of nitrogen, that is, it readily forms a compound of titanium and nitrogen. The valuable property is claimed for the alloy of removing the last traces of FeO from steel, and also of removing a large part of the absorbed nitrogen. Many tests have been made that seem to demonstrate the ability of ferrotitanium to reduce the FeO and remove the nitrogen of steel very thoroughly, and the result of its use appears to be a clean, strong metal. In the manufacture of steel for rolled shapes, the segregation of carbon, sulphur and phosphorus in the ingots is considerably reduced by titanium, resulting in superior toughness in the steel from the upper parts of the ingots. Segregation, however, is seldom an evil in foundry practice, as it takes place chiefly in the sink heads.

Whatever "wash" be used to reinforce the action of ferromanganese, it should be added after the latter has had an opportunity to perform as much deoxidation as possible. The "wash" being a more active deoxidizer than manganese, if added before the latter, or at the same time with it, will be wasted in performing the work that manganese can do, and none will be left over for the desired

¹ Trans. A. I. M. E., 1910, p. 803.

final deoxidation. If the recarburizers are added in the ladle, the "wash" should be added last, when the ladle is partly filled, and of course the same applies to recarburizing in the vessel or furnace.

The Ladles.—The ladles and large "shanks" used in pouring steel are made of steel plate appropriately bolted together and braced; small shanks are often cast in one piece, in the shop. Shanks may be carried by hand, or suspended by means of a "bail" from an overhead trolley or crane. They are poured over the lip by tipping. Small ladles are generally hung from a bail and provided with a hand wheel, rack and pinion, so that they can be revolved on their trunnions for pouring over the lip, or in order to turn them upside down and dump them. They may also be provided with a nozzle. Large ladles, holding over 5 tons or so, are generally poured through one or more nozzles and are not arranged to turn over. They may be picked up by heavy crane-hooks under the trunnions, or provided with a bail like the smaller ones. In any case, they should not be so hung as to be top-heavy when full, lest failure of the catch to engage allow them to dump themselves suddenly and disastrously.

The ladles are lined with refractory material to a thickness sufficient to protect the metal of the ladle from the heat of the steel. The thickness of the lining, therefore, varies with the capacity of the ladle. Small shanks, and ladles of a capacity up to some 3 tons, are commonly lined out by daubing a coating of ganister over the plating. The coating is made quite thick at the bottom and thinner toward the top. For a 3-ton ladle, the lining in the bottom should be about 6 in. thick, and may be reduced to 2 or 3 in. at the lip. In a small ladle, too heavy a lining is often worse than too light a one. because it is almost impossible to heat a thick lining thoroughly to a depth of over a few inches, and a heavy mass of refractory material at a comparatively low temperature in contact with the steel absorbs so much heat that for some time after the ladle is filled it acts to cool the metal rather than to keep it hot. The same thing applies to the linings of shanks and hand ladles; too heavy a lining often cools the metal rather than keeping it hot. Large ladles are lined with one or more courses of clay brick; those on the side are generally laid with the medium dimension of the brick in the line of the radius of the ladle. Over the bricks in the bottom is spread a layer of ganister, which is brought a little way up the sides. Smaller ladles also may be so lined, but the difficulty of removing a heavy "skull" from a small, brick-lined ladle is so great that the ganister lining is to be recommended. Small skulls in lip-poured ladles. especially in Bessemer work, are often left in the ladle to be cut out by subsequent heats; but a skull should not be left in a cold ladle.

Most foundries pouring manganese steel use as a ladle lining magnesite mixed with a small amount of clay. Enough water is used to make the mass moist enough to stick together, and it is rammed into the ladle with broad-faced steel hammers. The usual ganister linings are most unsatisfactory for manganese steel, because the manganese oxide forming on the surface of the bare metal reacts with the silica of the lining and forms a watery slag; constant skimming is then required to keep the metal clean, and it is almost impossible to keep the slag from getting into the moulds. For a somewhat similar reason, stoppers and nozzles cannot be used to pour manganese steel; the nozzles cut out so fast that after one or two castings are poured it is impossible to shut off the stream. In many cases, too, the sleeve bricks are cut through and the stopper rod burned off. Lip pouring, therefore, is standard practice for manganese steel.

Some foundries use the coffee-pot ladle with success for manganese steel and thus secure the advantages of lip pouring without being obliged to skim the slag from the ladles. This type of ladle, however, is not always satisfactory, especially when much heavy work is to be poured, for which it is necessary to allow the steel to grow quite cold. The spout of the coffee-pot is almost sure to choke up when pouring this cold metal, and the cost of ladle repairs is consequently very high.

The ladles for small acid electric furnaces are often provided with a "skimming lip" built into the lining, for the purpose of holding back the slag. This arrangement is similar to the coffee-pot ladle, except that the partition between the lip and the main portion of the ladle extends downward only 6 or 8 in. For steels covered with a viscous slag these lips work exceedingly well. A similar device is often used on the "bull ladles" in these shops.

In a small Bessemer foundry using lip-poured ladles, the author once saw a strong, well-wired dry sand core about 2 in. square by 10 or 12 in. long used to hold back the last of the slag after the bulk of it had been skimmed off. The core was floated on the steel at the lip, its ends touching the sides of the ladle, and kept back the small amount of floating slag on the metal very well. It could be readily put back in position with a skimmer if it floated out of place, and the small space in front of it remained perfectly clear of slag.

The lining of the ladles should be thoroughly dried out, either with

an oil or gas burner, or with a wood fire. When dry, the lining is heated up before use, either with the burner, or by inverting the ladle over a coal fire provided with forced draught. Large ladles, of course, cannot very well be turned upside down for heating, and must be dried with a burner. Small ladles and shanks are generally made at least red hot before use; large ladles need not be made so hot, though it does no harm, especially in case of a rather "cold" heat of steel.

The only feasible method of pouring large ladles is by means of a nozzle and stopper. To ensure satisfactory pouring, two nozzles are generally provided, so that if one gets plugged up the other can be used. Small ladles, of 1 to 3 tons capacity, generally have but one nozzle and are poured over the lip in case of trouble with the nozzle.

Nozzles may be of graphite and clay, or of clay alone. They should be put in place with care, and the lining rammed in around them thoroughly. The nose of the nozzle should project well beyond the plate used to hold it in position, so that the steel from a dribbling nozzle shall not freeze on the plate.

The stopper head is generally made of graphite and clay. Sometimes it is threaded to screw on the end of the stopper rod; but this design is a poor one, because the expansion of the rod in heating up is very apt to crack the stopper head. A far better design is the head with a hole clean through it for the reception of a bolt, which enters a hole in the lower end of the rod and is keyed in place. The hole in the stopper over the head of this bolt is filled with ganister after the stopper has been fastened to the rod.

In the proceedings of the American Foundrymen's Association will be found drawings showing the standard sizes and styles of stoppers and nozzles adopted by that body. As these designs are the result of much study, and are based upon years of experience, they should be followed whenever possible.

Nozzles with a deep seat for the stopper head, and in particular a steep-sided seat, should be avoided like the plague. Especially in large ladles, the stopper often gets slightly out of line with the nozzle, and with this steep-sided seat, the stopper then cannot "find" the opening. The best design is the round-nosed stopper and the nozzle with a seat sloping quite gently from the central opening to the outside surface. This type seldom gives trouble by refusal to seat properly.

Sleeves of clay brick, each with a tenon to fit inside a mortise in

the one below, are slipped over the rod. All joints in the sleeves are smeared with clay before putting the sleeves in place. To hold them tightly in position, washers are put over the uppermost, and a key driven through a slot in the rod. A thread and nut should never be used for this purpose, because the thread is sure to jam from heat and abuse.

Before placing a nozzle and making up a stopper rod for it, nozzle and stopper should be matched to see that they fit closely. This can be determined by pressing the stopper into its seat in the nozzle and turning it around. It should touch the surface of the nozzle at all points. If the fit is only slightly imperfect, the trouble can be corrected by pressing the stopper against the nozzle, and turning it back and forth till a good seat is secured. When the nozzle is so badly shaped that a considerable amount of such grinding is needed to secure a fit, it is better to reject the nozzle.

The stopper rod is fastened to the "goose-neck," which in turn is inserted in its socket on the ladle slide. Small ladles may be heated up with stopper rod and goose neck removed, and the rod dried out separately. For large ladles, it is best to leave the stopper rod in place, and merely raise the stopper off its seat while heating the ladle.

In open-hearth practice especially it is most desirable to follow this rule, because it is then easy to have the ladle ready at a moment's notice in case the heat has to be tapped because of a break-out or when, as sometimes happens, the heat comes through the tap-hole of its own accord. The author has seen an open-hearth crew stand helplessly by and watch 30 tons of good steel run into the ladle pit, simply because they made a practice of not placing the stopper rod in the ladle till they were about ready to tap; and on this occasion, the metal tapped itself before the ladle was ready.

In any case, the stopper should be carefully adjusted before the ladle is heated, so that it closes the hole in the nozzle exactly. To ascertain if the stopper leaks, it is seated firmly and a little fine sand thrown around it. This testing with sand should be repeated when the stopper is put in place or lowered into position, after heating the ladle, and a ladle should not be filled until the ladle man is sure that it does not leak.

Some ladle men prefer, especially for large ladles, to arrange the stopper so that it strikes the outside of its seat in the nozzle, and slides into position as it is pressed home. It is generally best, however, to have the stopper come vertically down upon its seat.

In pouring through a nozzle, the rod should be raised slowly the

first time the nozzle is opened, and the pourer will do well, soon after he has got his stream going, to close off once or twice. By so doing he clears away from the nozzle any half-melted sand that may be sticking to it, and ensures a clean shut-off. The first few times the nozzle is seated after the first opening, it often will not shut off clean.

Should the stopper get stuck hard to the nozzle, it should be poked off with a pricker from below, rather than trying to pull it off with the ladle handle. The result of too much enthusiasm at the handle is sometimes that a stopper head pulls off the rod. A frozen nozzle in large ladles is pricked open in the same way. Wooden prickers are better for this purpose than steel ones, as they do not freeze to the metal in the nozzle; but a steel pricker often has to be used for a badly frozen nozzle. In pouring a small ladle, if the nozzle freezes up, the steel had better be poured over the lip.

CHAPTER VIII

MOULDING, POURING AND DIGGING OUT

The effect of moulding and pouring methods upon the soundness of the castings is so great that in order to be useful to a steel foundry. the metallurgist is obliged to be thoroughly familiar with the subject. Not that he need necessarily be a foundry expert, versed in the economies of moulding practice, the use of machines, etc., but as concerns the effects of different methods upon his steel he must be an expert. Many a foundry makes a practice of holding the steel maker responsible for blowholes, hot checks, shrinkage cavities, etc., in the castings, when the trouble is really the result of ignorant or careless foundry practice, and the melting shop men too often take the attitude that nothing is wrong about their metal, which would pour into the most beautiful castings ever seen, if only the foundry were in charge of men who knew their business. To prevent this cat and dog attitude of foundry and melting shop, often calls for a degree of tact on the part of the superintendent that too few men possess. Frequently an attempt is made to cut the Gordian knot by giving the foundry superintendent charge of both moulding and steel making, and indeed when a man can be found who is well versed in the practice of both departments, no better arrangement Too often, however, the man who is an expert in handling the moulding, possesses but a smattering of the principles of steel making, and under his rule the lives of the melting shop men are made a burden to them.

Mixing the Sand.—The mould in which the casting is poured must retain its shape without spalling, fusing or softening when it is filled with molten steel; and it must carry away the gases that escape from the steel and that are driven off from the layers of sand close to the casting, so that these gases shall not be entangled in the solidifying metal and form blowholes. The sand, therefore, must have a high-fusing point, and it must be permeable to the flow of gases.

The first requirement is best fulfilled by a pure silica or quartz sand. In practice, most sands used for steel moulding contain by analysis at least 90 per cent. silica, some even as high as 98 or 99 per cent. Such a sand will have a very high-melting point, so that

none of it will soften when heated by the steel; if this occurs particles of sand stick to the metal so tightly that it is impossible to get them off.

Silica sands are either river or beach sands, or sandstones or quartzites artificially crushed. The river and beach sands are almost invariably composed of well-rounded grains, and often each grain consists of a single quartz crystal worn smooth by the action of the water. Crushed quartzites and sandstones may show a similar appearance under the microscope, or they may be composed of sharp, angular fragments. If these grains are each composed of portions of several crystals, the sand will tend to become finer in use, because the grains when heated by the steel will shatter into several pieces.

The shape of the individual grains of the sand, and especially their uniformity of size, are of great importance in determining the proportion of open space in any given volume of the material, and hence the permeability of the sand to the flow of gases. It is well known that a mass of spheres of uniform size will fill a given space and leave more voids between them than will any other shape, and that the larger the diameter of the spheres the greater the percentage of voids when the space is packed as full of the spheres as it will hold. A simple way to prove this would be to fill a small glass first with buck shot, then with No. 6 and then with No. 10 shot, weighing the amount of each size required to fill the glass, and comparing that weight with that of a chunk of lead of the same dimensions as those of the glass. Then fill the glass with a mixture of these different sizes, getting in all the No. 6 and No. 10 that can be shaken down between the larger grains, and again compare weights. The difference between the weight of a piece of lead just the size of the glass. and that of the shot needed to fill the glass, is a measure, of course, of the amount of voids between the grains, and will be found greatest for the buck shot, and least for the mixture of sizes.

The ideal material, therefore, would be one in which all the particles were spherical, all of the same size, and each consisting of a single crystal so that the grains would not shatter when heated. In such a sand, the amount of open space per cubic inch would be a maximum, hence the permeability would be the greatest possible. Actually, of course, we must work with such materials as we can secure, and the price must be low. It is almost always necessary, therefore, to obtain our sand from near-by sources; but often careful examination of the sand in use will lead either to the substitution of

some other source of supply, or to the improvement of the quality of sand obtained.

More than one kind of sand can now be bought that has been graded or sorted artificially and is furnished coarse, medium or fine. To determine the uniformity of grain size of the sand, a properly selected sample should be put through a standard set of sieves or screens, which have respectively 20, 40, 60, 80 and 100 meshes or spaces between wires, to the inch. The amount remaining on each screen and that passing through the 100-mesh screen, is weighed to determine the proportion of each to the whole sample. What remains on the 80- and 100-mesh screens is very fine, the part that goes through 100-mesh is practically dust. It is, of course, undesirable to have a high proportion of this finer material in our sand, as the fine particles choke the openings between the larger grains, and make the sand "close." The larger the amount remaining on the 20-, 40- and 60-mesh screens, the better the venting properties of the material; some foundrymen prefer coarse sands, in which the greater part remains on the 20- and 40-mesh screens, others get the best results from finer material, the greater proportion of which remains on the 40- and 60-mesh. Whichever end of the scale is preferred, the greater the uniformity of grain size, the better.

In the following table, taken from a most interesting paper by the late R. L. Lindstrom, are given the screen tests and silica contents

| | | Fineness | | | | | Total | Silica | |
|-----|--------------|----------|------|-------|------|------|-------|--------|-------|
| No. | State | Mesh | Mesh | Mesh | Mesh | Mesh | Mesh | on 60 | con- |
| | | 20 | 40 | 60 | 80 | 100 | 100X | mesh | tent |
| 1 | Illinois | 10.75 | 33.6 | 44.02 | .37 | 2.41 | 5.62 | 88.34 | 97.25 |
| 2 | Pennsylvania | 8.00 | 81.0 | 7.5 | 1.5 | - 5 | 1.5 | 96.5 | 97.25 |
| 3 | Ohio | 19.0 | 29.0 | 33.0 | 14.0 | 2.5 | 2.5 | 81.0 | 97.40 |
| 4 | Ohio | 29.2 | 51.8 | 12.2 | 3.4 | .9 | 2.5 | 93.2 | 96.52 |
| 5 | New Jersey | 11.0 | 17.4 | 25.6 | 19.9 | 10.2 | 15.0 | 54.9 | 97.00 |
| 6 | Ontario | 2.4 | 62.8 | 26.4 | 6.9 | .7 | .8 | 91.6 | 98.97 |
| 7 | Quebec | 1.6 | 25.8 | 33.I | 27.4 | 5.6 | 6.5 | 61.5 | 97.70 |

of a number of typical moulding sands. Attention is particularly called to the variation of the average size of the grains in the different sands; thus of sand No. 2, 81 per cent. remains on the 40-mesh screen, and 96.5 per cent. is coarser than 60-mesh, while No. 5 varies so greatly in size that but 54.0 per cent. remains on the 20-, 40- and 60-mesh screens.

The accompanying illustrations Figs. 12 to 15 show the grains of sands Nos. 1 and 7, in the 40- and 100-mesh samples, magnified 20

¹ Lindstorm, R. L., Proc. Am. Foundrymen's Assoc., 1920, vol. xxix, p. 202.

diameters. Number 1 is probably the best steel moulding sand in this country, as is clear from the shape of the particles, and the uniformity of grain size. Number 7 gives a fair screen test, but the



Fig. 12.—Sand No. 1—40 mesh. Fig. 13.—Sand No. 1—100 mesh. Both \times 20 diameters.

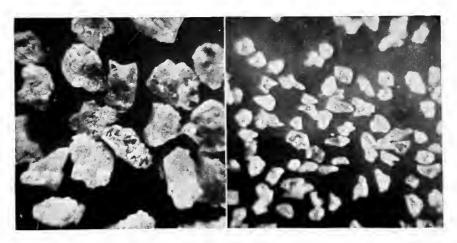


Fig. 14.—Sand No. 7—40 mesh. Fig. 15.—Sand No. 7—100 mesh. Both × 20 diameters.

microscope shows at once that the particles are of such irregular, angular shape that they pack too densely together; the worst feature of this sand, however, is that the grains are each composed

of portions of several crystals, so that when heated by the steel they split up into small fragments, increasing the fineness and decreasing the permeability.

Pure quartz sand, of course, has no bonding power, so that some substance has to be mixed with it to hold it together when rammed into the moulds. The best material for this purpose is clay, which is nearly always present in the sands of lower silica content, in sufficient amount to provide the required bond. When the silica of the sand is over 95 per cent., clay generally has to be mixed with it before it can be used for moulding purposes.

The clay occurring naturally in the sands of low silica content, and that used for the purer sands, should have a high-melting or softening point, and the greatest possible bonding power. The high-fusion point, of course, is desirable to prevent the clay from softening under the heat of the steel and causing the sand to burn to the castings. The bonding power should be high so that as little clay as possible need be used to make the sand sufficiently strong; for if a great deal of clay is mixed with the sand, it will fill the interstices between the grains and destroy the permeability or venting qualities. Strangely enough, the properties of any given clay cannot be predicted from the analysis, the bonding power in particular varying to a marked degree. It is desirable, therefore, to choose a clay entirely by test, which may be either a service test or a laboratory examination, or both; preferably both, as in the case of a poor material, testing only by actual use may be very costly.

In the paper referred to above, Mr. Lindstrom states that the softening point of a good steel-foundry clay should be not less than 1600°C., and he gives the details of a simple test for bonding power. This consists of mixing a fixed proportion of the clay with clean, high silica sand, making of this mixture a slab of given size, and pushing the completed slab over a clean glass plate until the portion overhanging the edge of the plate breaks off. The length of the portion breaking off is taken as the bonding strength of the clay. The kind of sand with which the clay is mixed, the amount of water in the mixture, the length of time the batch is worked, and the pressure put on the slab, must all be kept constant. This or some similar test, coupled with the determination of the softening point, should be of great value in selecting clays for actual trial in the foundry.

If the new sand coming into the foundry were of uniform analysis and fineness, and if only new sand were used in the moulds, the

foundryman's problems would be greatly simplified. Unfortunately the sand received often varies considerably from car to car, and considerations of economy compel us to incorporate considerable proportions of old sand in each batch mixed. The presence of the old sand in the mixture complicates the problem of getting the bonding properties and openness just right, partly because some of the grains of old sand have split up, and partly because the nature of the clay in the old sand has been changed by the heat of the castings. Exposure to considerable heat alters the physical condition of the clay in such a manner as to destroy or greatly diminish its bonding power; and as the depth of the layer of sand in which this change has occurred varies with the mass of the casting, it is practically impossible to know just how much new clay must be added to restore the bond in the old sand, as the proportion of burned clav in the latter varies widely from day to day, and from one part of a heap to another. It must be remembered too, that this clay whose bonding properties have been destroyed is still present in the sand, and fills up the insterstices between grains just as much as does the new clay.

The exact proportions of old and new sand and of clay, therefore, must be varied from time to time, and the same proportions cannot be used for all classes of work. No mixture should be regarded as standard, but the foundryman should watch his sand and his castings closely, and vary the mixture, using less clay and old sand when the sand appears to be too close, and at times even doing away with the old sand altogether. In some few shops no "heap sand" is used, but the moulds are filled entirely with newly mixed sand. the average foundry, however, only the facing is especially prepared, and the greater part of the flask is filled with heap sand from the floors, which is made ready for use each say simply by mixing it with water and shoveling it over. It must not be forgotten that to these heaps is daily added a certain amount of sand in which the bonding power of the clay has been destroyed, so that as much heap sand should be removed from the shop each day as is added in new sand, not only to prevent the accumulation of too much sand, but to keep the heap sand as uniform as possible.

For "green-sand" moulding it is most important that the venting properties of the sand should be as nearly perfect as possible. The best green-sand work is done with nearly pure silica sands of uniform, rather coarse, grain, mixed with the minimum of the best clay, and with as little water as possible. A handful of the facing

given a single hard squeeze in the hand should just hold together and no more. Too much moisture in green-sand moulds, of course, gives rise to an excess of steam, which cannot escape through the close rammed moulds and causes blowholes in the steel. The floor sand, therefore, as well as the facing, should be kept as dry as it can be used without falling out of the flasks. In some cases, too. where the sand of the mould resists the contraction of the casting as it cools, the sand must not be rammed too hard or it will not yield and as a result the casting will crack. In extreme cases the writer has seen sea-coal (finely ground bituminous coal) mixed with the sand in such parts of the mould. Its function, of course, is to burn out as soon as possible after the casting is poured, and thus loosen In these cases, however, it is necessary to be perfectly sure that the part of the mould where sea-coal is mixed with the sand is perfectly vented; were it not, some of the great volume of gas given off by the coal would assuredly find its way into the casting and form blowholes.

Green-sand moulds are frequently "skin dried" with a blow-torch, or by spraying them with gasoline and setting it afire. The object of this, of course, is to drive out a large part of the water in the layers of sand close to the casting, and so minimize the amount of steam arising from the sand during and immediately after pouring. As the water in the rest of the mould soon migrates by capillary action and takes the place of that driven off in skin-drying, the moulds should be poured as soon as possible after drying.

For "dry-sand" work more clay can be mixed with the sand than is allowable in green-sand practice, as the mould is dried before pouring and much less steam is generated in the moulds when they are poured and such perfect venting is, therefore, not essential. Molasses water or some other binder is often mixed with the facing to help hold it together, in addition to the clay. For dry-sand moulding the sands containing a considerable proportion of clay in the natural state are extensively employed, and give excellent results. The facing sand, even for moulds that are to be dried, should not be too moist, as it is then prone to flake and spall in drying, rendering necessary a good deal of patching in setting up the mould. If the patches are not dried out, and especially if the silica wash be applied to them too freely, the resulting wet spot may cause blowholes in the casting.

The heap sand should not be too wet, even in a dry-sand mould, as very damp sand tends to pack so hard as to be impervious to the

escape of gases, even when thoroughly vented; and a wet sand results in greatly increased drying time, or in a mould brought from the oven and poured with damp spots in it. A mould impervious to the escape of gases, especially if it is wet in spots, is sure to cause blowholes in the steel.

Cores and Core Sands.—What has been said of moulding sands applies even more strongly to core sands, because of the greater necessity in the latter for perfect venting. The mould proper surrounds the metal of the casting, so that the gases from the steel and those generated in the sand, can escape outward through the mould; indeed during pouring, quite a quantity escapes into the unfilled portion of the mould and is taken off through "flow-offs" or sink heads. A core, on the other hand, is a body of sand more or less surrounded by the metal; in many cases a very large proportion of the core is imbedded in the casting. The gases generated in the core, therefore, must escape through the core itself, otherwise they will have no place to go but into the solidifying metal around the core where they will form blowholes. The venting properties of the core sand, therefore, must be nearly perfect.

Moreover, the core while submerged in the fluid metal is relatively in the same position as a block of wood held beneath the surface of a body of water; being far lighter than the steel with which it is surrounded, it tends to float upward. It is absolutely vital, therefore, that the core should be strong, else it will surely break and float as a unit, or fragments will spall off and ruin the casting. Steel rods or iron "arbors" moulded into the cores to a large extent provide the requisite strength of the core as a unit, but even so the necessity for using a strong core sand remains.

To obtain the necessary strength in the core sand with clay alone as a bond is out of the question, because so much clay would be needed as to fill up the voids between the sand grains and destroy the venting qualities of the core. Some other binder, therefore, must be employed, that will give the necessary strength without unduly decreasing the venting properties. Moreover, it is essential that the core lose its strength after the metal has solidified around it, partly so that the sand of the core may be easily dug out, and partly because in many cases if the core remains hard and unyielding, it will crack the casting about it as the latter contracts in cooling off. In large cores only the outside portion is of core sand, the interior being filled with cinders or some other loose material, whereby the core is made to yield more readily as the casting contracts,

especially when as is often the case, the core is broken up with bars or picks after the casting has solidified. The loose cinders in the interior of the core serve also to provide the necessary vent to carry off the gases from the sand with which the core is faced.

In general, core binders may be liquid or dry. Of the former linseed oil, molasses, corn sugar syrup and concentrated waste sulphite liquors from the manufacture of paper pulp are the most used. The dry binders include flour, various gums and pitches used in black compounds and products such as dextrin derived from certain of the cereals. The liquid binders tend to migrate to the surface of the cores during baking, so that they are not well suited to heavy cores, since during the long sojourn in the oven necessary to dry these large masses of sand, too much of the binder reaches the surface, resulting in a core with a weak interior and a hard impervious surface. Cores made with the liquid binders, also, have a greater tendency to absorb water and soften if left long in the moulds or in a damp place, than those made with the dry binders.

For a detailed discussion of the properties of the various binders suitable for core-making the reader is referred to a small volume entitled "Core Making," by Dr. Richard Moldenke, and printed by the Corn Products Refining Company, in which will be found a great deal of valuable information on the subject.

In selecting core sands, the same general principles should be followed that have been indicated in the discussion of moulding sands. For small cores where perfect venting and considerable strength are essential, the pure silica sands of rather coarse grain are best, and very little if any old sand should be incorporated with the new material, on account of the difficulty of controlling with certainty the amount of binder in the batch. Linseed oil is extensively used for these small cores, though other binders are satisfactory. Such cores are provided with a vent hole from end to end, which can be made with a wire; but many foundrymen consider ventwax, which of course melts out when the cores are dried, the most reliable means of ensuring a perfect vent.

Larger cores are made with more or less old sand, and with such a proportion of binder as will give the necessary strength without making the core too hard; for the foundryman must ever be on the alert not to get his cores so hard as to result in cracked castings.

Cores that are to be surrounded with a heavy body of metal, too, must be heavily coated over with some substance such as a wash of silica flour, so that when the binder is driven out from between the grains of sand by the heat of the metal, the steel shall not penetrate between the grains and form a sort of steel-and-sand pudding stone that is almost impossible to chip out. A considerable proportion of clay as a binder for such cores, or silica flour mixed with the sand will serve the same purpose, but here again care is necessary lest the core be too hard to yield as the casting cools and shrinks.

Venting.—When steel is poured into a mould, great volumes of steam and gas are given off, formed by the vaporization of the water and binder in the mould and cores. As the mould fills up, these gases must be allowed to escape. Generally, the sink heads take care of the greater part of the gases given off during pouring, but frequently small flow-offs or "whistlers" must be provided to carry away the gases from portions of the mould where risers are not required for any other purpose. In most cases, however, it is not the gases evolved during pouring that form blowholes in the castings, but those coming from the sand and those given off by the cooling steel, after the mould is full and before the steel has solidified. To take care of these gases, the vents from cores and from portions of the mould projecting into the casting, must be brought out through the mould so that they will carry away the gases freely. For the same reason the copes of small moulds are vented by driving a long needle or bodkin through the sand in numerous places before the pattern is removed from the mould.

No matter how careful the moulder is in providing vents, however, they will be ineffective unless the sand itself is so proportioned as to vent freely; for if the sand be impervious, the vent holes will carry off no gas, because none will flow through the sand to reach them. The selection and preparation of the sand, therefore, are of preponderating importance in the problem of how best to carry off the gases that form after the mould is filled, so that they may not force their way into the solidifying metal and form blowholes.

Defects in Castings.—The foundryman has to guard against several types of imperfections in his castings, which can be divided into three classes as follows:

- 1. Blowholes, sand holes, scabs and other exterior imperfections.
- 2. Shrinkage cavities due to contraction of the steel, and formed chiefly during solidification.
- 3. Stresses set up and cracks formed by contraction of the steel in the solid or semi-solid state.

Blowholes in castings may be due to badly made steel, in which case most of the castings poured from a heat will be affected. Should

it be found that only certain castings contain blowholes, it is clear that the sand used and the moulding practice followed have been at fault. In the discussion of moulding and core sands in the preceeding pages are set forth the principles that make for success or failure as far as blowholes are concerned.

Sand holes and scabs are caused by the flaking off of a certain amount of the mould, due to wet spots, too tight ramming, insufficient binder and other causes. Patches made in dry sand moulds. too, frequently spall off during pouring and produce scabs and bad spots on the surface of the castings. At times insufficiently refractory sand will be found responsible for an unsightly surface, as it burns to the steel and is then almost impossible to remove. Slag carelessly allowed to enter the mould in "lip pouring," too, results in surface imperfections, which sometimes are difficult to distinguish from sand holes. Insufficient nailing of copes and of the bottoms of sprues and other places where the steel is apt to cut the mould in pouring, and carelessness in blowing out loose sand before closing a mould are other causes of sand holes in castings. For the benefit of those who are not conversant with foundry practice, it should be explained that to "nail" a mould is to push into the sand a number of wire nails with comparatively large heads, at intervals of 1 or 2 in. The nails are shoved all the way in, the heads remaining on the surface of the mould serving to keep the sand from washing away or spalling off.

Lastly, when the steel is poured too cold imperfections known as "cold shuts" may occur. These are places where two portions of the flowing steel have met in the mould, as for instance as the metal flows over or around a small core and meets again on the far side, and the metal has been too cold for the two streams to re-unite. These defects are not often very serious, but the worst of them is that it is almost impossible to determine by surface inspection whether they are mere wrinkles in the skin, or whether they extend to some depth.

Shrinkage cavities as the foundryman terms them, or "pipes" as they are called when they are found in ingots, are formed during the solidification of the steel, and unless means are taken to prevent, some portion of every casting will contain such a cavity. The ingots used for rolling and forging are always more or less hollow at the top, and much careful study has been devoted to discovering the best means of minimizing the size of these "pipes" or shrinkage cavities, as well as to discovering the mechanism of their formation. As

ingots are the simplest form into which steel is cast, a study of what has been learned about them is of great benefit to the foundryman; for if one understands the mechanism of "piping" in the ingot, a perfectly plain section, one can reason intelligently in endeavoring to make sound castings of complicated form.

Prof. Henry M. Howe and Bradley Stoughton have written two most illuminating papers¹ on this interesting subject, that will amply repay the foundryman for the labor of reading. To condense such a paper as that on "Piping and Segregation in Steel Ingots" is almost impossible, but as the explanation of the formation of shrinkage cavities given in that paper is fundamental, the following summary is attempted.

It has not yet been definitely established whether steel actually expands or contracts at the moment of solidification. Whichever it does, however, the inevitable result of the comparatively rapid cooling of a mass of steel like an ingot, is that as the metal freezes around the outside of the mass, and the shell of solidifying steel gradually thickens, the liquid metal in the interior, after a certain stage of the cooling is reached, fails to fill completely the hollow enclosed by the steadily thickening walls, and from that moment until the last drop solidifies, the amount by which the liquid metal fails to fill the solidified shell steadily increases so that the level of the liquid metal gradually falls. The result is the familiar pear-shaped opening in the upper part of the ingot, known as a "pipe."

The explanation of this is as follows: Consider the shell of steel frozen around the liquid ingot as consisting of two parts, the outer layer and the inner layers. The steel of all the layers tends to contract as it cools. The outer layer cools much faster than the inner layers, until it has reached comparatively low temperatures. Hence this outer layer tends to contract more rapidly than the layers within it. The layers, being actually parts of a continuous mass, cannot slide upon each other in contracting. Hence, the contraction of the fast-cooling outer layer is in part prevented by the resistance to its motion of the slower-cooling inner layers, and the outside material is "virtually expanded" because the stresses set up in it by its contraction against the resistance of the inner layers exceed its elastic limit and cause the steel to yield. Thus in the first period of cooling, which Professor Howe calls the pipeless period, the outer walls of the ingot are held out to a length which they should not

 $^{^1}$ Howe and Stoughton, *Trans. A.I.M.E.* 1907, p. 109; Howe, same volume, p. 3.

properly possess, and would not, were their contraction unimpeded; and when they are cold they are longer than they would have been had they been allowed to contract freely.

As the outer layers become colder and colder the heat is abstracted from them at a diminishing rate, and finally a point is reached when the inner layers are cooling more rapidly than the outer. The second, or pipe-forming, period then begins. During this period, the inner layers are cooling and contracting faster than the outer, to which they are integrally attached. Hence the contraction of the inner layers toward the outer increases the size of the interior cavity occupied by the molten metal, and the latter fails to fill the cavity. Therefore, its surface sinks down more and more as the freezing walls of the ingot thicken and the size of the interior cavity The result is the familiar pear-shaped cavity, with small vertical ribs or corrugations. Below this smooth-sided portion of the pipe is a narrow, steep-sided and rough-sided portion resulting from the tearing apart of the solid but soft metal below the part of the pipe formed during actual solidification, this tearing apart being due to the continued outward contraction of the inner layers of the ingot as the cooling of the solid metal progresses.

The depth of the pipe is lessened:

- 1. By casting in wide ingots.
- 2. By casting in sand-lined moulds (particularly if the moulds are pre-heated).
 - 3. By pouring from the top instead of from the bottom.
 - 4. By pouring slowly.
 - 5. By casting large end up.
 - 6. By casting with a sink head at the top.
 - 7. By permitting blowholes to form.
 - 8. By liquid compression in presses.
- r. The use of wide ingots so increases the total amount of heat to be abstracted from the steel and carried away by the mould walls, as to decrease greatly the proportion which the length of the first period of cooling, the pipeless period, bears to that of the second or pipe-forming period. Hence the relative amount of the virtual expansion of the outer layers is lessened, and the tendency to deep piping diminished. Thus Professor Howe explains the formation of the very deep pipe found in narrow ingots.
- 2. The use of sand-lined moulds decreases the rate at which heat is abstracted from the exterior layers when they are first frozen. If the mould is pre-heated, this is even more pronounced. Hence, the

amount by which the exterior layers outstrip the inner layers in cooling is decreased, the virtual expansion of the outer layers diminished, and the tendency to deep piping lessened. The most obvious conclusion to be derived from this is that castings generally pipe less deeply than ingots.

These two methods, the use of wide ingots and the use of sand lined moulds, act by diminishing the tendency to deep piping.

3 to 6. Methods 3 to 6 are pipe-closing in their tendency, and act by favoring what Professor Howe has called "down sagging," that is the sliding of the plastic, semi-solid metal upon the interior of the pipe, down its walls. This sliding downward of the metal upon the inside of the pipe as the walls of the ingot thicken will manifestly tend to fill the bottom of the deepening pipe, and hence to limit the depth that it will finally attain. In order that this down-sliding should be a maximum, the metal that is to slide manifestly must be in a position to do the most sliding after the pipe begins to form. Hence, it must do most of its solidifying after the pipe has begun to open. It is an obvious inference that the upper metal, that is to slide downward, must freeze after the metal below. Therefore, the top of the ingot should freeze last. Pouring from the top has this tendency because it fills the top last with hot metal from the ladle, which is run in above that already in the moulds, whose cooling has already begun. Pouring slowly allows the cooling of the lower portion of the ingot to proceed very far before the hot top metal is poured. Pouring large end up, by increasing the amount of heat to be abstracted from the top of the ingot and decreasing that to be taken from the bottom, has the same effect of keeping the top hot and favoring the maximum down-sliding. A sink head, especially if cast in sand so that it cools more slowly than the rest of the ingot, of course has the same effect.

7 and 8. Permitting blowholes to form decreases the depth of the pipe by forcing the interior metal to occupy more space than it would if sound, and so to expand and rise into the pipe. Fluid compression, properly applied, also forces the interior metal into the pipe. The latter method is of no interest to the foundryman, for obvious reasons; and as blowholes are highly undesirable in the average steel casting, it does not help the foundryman much to know that they will aid him to decrease the size of the shrinkage cavities in his castings.

In applying to castings the methods of decreasing depth of pipe above outlined, we see at once that the first two methods, the use of wide ingots and of sand moulds, are not directly applicable; the first because the thickness of our sections is fixed by the design of the casting to be made, the second because we cast in sand anyway. There are, however, certain very practical applications of these two methods. The first is perhaps more closely allied to No. 5, casting large end up, and is that as far as possible we should try so to place our casting in the mould that heavy sections are above light ones. Thus we ensure that the upper portions will tend to freeze last, and so feed the contraction of the lighter sections below to the greatest possible extent. From Professor Howe's explanation of the causes of piping, we see that the light portions of a casting are, per se, more prone to the formation of pipe and shrinkage cavities than the heavy sections. In a great many cases this tendency is masked by the fact that these light portions are integrally united to heavy parts of the casting, which freeze last and contribute their interior metal to fill the cavities forming in the light portions. The drain thus brought upon the heavy sections, added to their own tendency to pipe, results in their being more often found hollow than the light sections, and so we say the heavy sections are most prone to contain pipes and shrinkage cavities.

In cases where the casting is poured with the light sections above the heavy ones, we frequently find the former badly affected, a great part of the opening being in the form of rough-sided cavity. We say we are dealing with a shrink-hole. In many cases, no doubt, this is the lower portion of the pipe, drawn open after the metal has frozen, and much exceeding the true pipe in size. Had there been a means of feeding the section, the cavity would have been filled.

Sink Heads and Chills.—To fill these cavities in the upper portion of castings, we provide sink heads to serve as reservoirs of liquid metal which will still be molten after the pipe in the casting has begun to form, and contribute their contents to fill the pipe, the sink heads being largely hollow when they finally freeze; and we use heavy chills, which are plates of iron or steel imbedded in the mould in such a way that the portions of the casting that we wish to cool first will be in contact with them. The function of these chills is not, as in iron castings, to produce a change in the structure of the metal next to them by which it is made harder, but simply to cause the metal that comes in contact with them to cool rapidly. The object of this is twofold; first, it evens up the rate of cooling of a casting with light and heavy sections, and thereby lessens the

tendency to pull apart when the light section solidifies and contracts and tears away from the still soft and weak heavy section; second, it causes part of the casting to cool and solidify rapidly, often while pouring is still in progress, so that in that part of the casting contraction occurs and the pipe or shrinkage cavity is formed, while other parts of the casting are still molten. As a result of this early solidification of a part, generally the lower portion, of the casting, due to the chill, the feeding of that part takes place earlier than it otherwise would; in this way we ensure that the other portions will remain molten longer than the chilled part, and hence that the feeding will be thorough. Moreover, as the feeding occurs thus early, part at least of the metal required to fill the cavity will be provided by the metal being run into the mould, instead of entirely by the overlying portions of the casting and the sink heads, and thus we may be able to make the casting solid with a smaller head than we would have to use did we not provide the chill. Indeed, cases frequently arise in which a casting of complicated shape cannot be made sound with sink heads alone, but can be successfully produced if chills and sink heads are properly placed and proportioned.

To be really effective, chills should be pieces of iron or steel bedded in the sand of the mould, heavy enough to have a pronounced chilling effect upon the metal that comes in contact with them, and what is often more important, heavy enough so that they are not heated nearly to their melting point and thus do not "burn fast" to the casting. Attempts to prevent a thin chill from sticking to the casting, by means of oil or grease, are usually futile; the best "chill grease" is an increase in the thickness of the chill, though a heavy coat of silica flour wash is helpful.

Some foundrymen attempt to secure the effect of a chill by means of nails stuck into the sand of the mould, and left extending out into the space to be occupied by the part of the casting they wish to chill. In extreme cases, the author has seen these nails placed so closely together that perhaps 20 to 30 per cent. of the volume of the casting at that point was composed of nails. This practice cannot be too heartly condemned. If enough nails are used to have a proper chilling effect, they cannot possibly melt and unite with the steel of the casting, but are merely "burned" in, and weaken the metal greatly; and even when so few nails are used that they really do no good, they seldom melt when the metal flows around them, and thus not only serve no useful purpose but are a positive source of harm.

Gates.—To return to the discussion of the prevention of pipes;

theoretically the best way to fill a casting is from the top (Method 3), straight down the sink head, filling the top slowly (Method 4). heavier sections should be above the lighter, that the top may freeze last and feed the bottom (Method 5); in some cases the bottom sections should be poured in chills (Method 2, reversed); and there should be extra metal above the casting proper in the form of a sink head (Method 6), to feed the contraction of the upper portions. By such a method of pouring we should use all the available means for obtaining sound ingots. The stream of metal falling into the mould, however, would necessarily cut the sand so badly as to ruin the casting; or if cores intervened, they would be destroyed. Hence, we are compelled in the majority of cases to take the metal down a runner exterior to the casting itself, and into the casting by means of gates at or near the bottom. By dishing out the bottom of the runner and "nailing" the part so dished, we provide a basin whence the metal can enter the mould quietly and without cutting the sand, or spattering in such a way as to lead to surface imperfections.

Our moulds, therefore, are commonly filled from the bottom, so that the metal which finally forms the top of the casting is that which first entered the mould and is the coldest. This is in direct violation of principle 3, but in most cases it is the only way to do the pouring. True, it is quite common to stop pouring through the gate when the metal reaches the sink heads, and pour the latter from the top, and this practice is theoretically correct, (Methods 3 and 4). But in some cases this is not sufficient, as the cooling of the upper part of a tall casting may outrun that of the bottom portions, poured last, to such an extent as to make it impossible for the feeding of the sink heads to be effective. We may easily imagine a case where the top metal is so cold when pouring stops that it freezes completely, forming only its own small pipe. The sink head will feed this; but the later-freezing lower portions will form their own contractioncavity, which is completely bridged off from the sink head by the frozen upper portions.

This condition may often be met by arranging a series of gates at several levels, so that as soon as the mould and runner are filled up to the level of gate two (counting from the bottom) the metal will enter the mould through this gate and run in over that already in the mould. When the steel reaches the level of gate three, the process will be repeated. In this way we attain the advantages of top pouring, at least in part, by pouring in a series of sections; each section as it were acting as sink head for the part below, and the

sink head proper feeding the last section poured. This is the first advantage of the use of gates at several levels. We shall again refer to the subject and mention the second advantage, in discussing the question of the size of gates and runners.

There are certain disadvantages of this method, which may in some cases prevent its use. The first is that any loose sand from the mould, which ordinarily floats upon the steel as the mould is filled, and hence is carried upward to the sink head where it can do no harm, may be trapped in the casting when the metal begins to flow in through the upper gates. The second is the danger of filling the runner too quickly, so that metal will flow through the second gate before the metal in the casting has reached that level, and cut out the mould as it flows down its walls. These disadvantages, however, are in many cases neither insurmountable nor prohibitive.

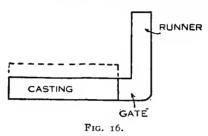
In certain cases where the vertical height of the casting is small compared to that of the runner, and no separate sink head is used, the runner itself may be considered as the sink head. In this case the action of properly placed chills may cause the feeding of the casting by the metal being run in, to proceed so far that the metal in the runner will complete the work after pouring ceases and make a perfectly sound casting. The gates and chills, of course, must be so arranged that the later-poured metal may flow in over that already in the mould; hence the neighborhood of the gate must not be chilled. It is also essential that the first metal be run in rapidly, and the pouring completed slowly.

In some cases the runner will feed such a casting, even without the assistance of chills, in others the chills are necessary to produce a sound casting; and in any case, the casting must be low compared to the runner, so that there is no considerable portion of it above the gate. In theory, at least, such a casting, in order to be made sound by the metal in the runner alone, should be provided with a gate from top to bottom on one side, so that the feeding of all levels of the casting is by metal running in from above. Actually, of course, the gate need not be quite the full height of the casting, as shown by the dotted lines in Fig. r6.

Even when a sink head is used in addition to the chills, so that we do not depend upon the runner to feed the casting after pouring ceases, this action of the chills is valuable, since it effects the feeding of the lower portions of the casting by the metal being run in through the gate. To take advantage of it to the full, the sink head should

be poured separately and from the top. Otherwise, we shall fill the head with the first-poured metal that has been cooled by the chills, and the head will tend to cool before the casting.

We spoke above of the fact that a high casting, poured through a single gate at the bottom, may freeze at or near the top to such an extent as to bridge off the sink head from the lower part of the casting, with the result that the piece will be unsound below. This, we saw, can be guarded against by the use of gates at several levels. The same end may be attained in certain cases by the use of chills on the lower parts of the casting, which so hasten the cooling of those

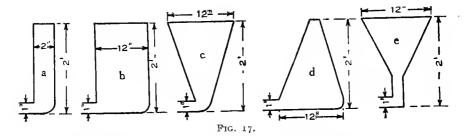


portions as to cause them to freeze ahead of the upper parts. Manifestly, however, it will be advisable in many cases to use both these methods, since their effects are cumulative. Indeed, if the metal is all poured through a gate at the bottom, the effectiveness of the chills will inevitably be reduced by the chilling of the metal that first enters the mould and ultimately forms the top of the casting, which has to flow over the chills to a certain extent and consequently is not as hot as it should be when it reaches its final position. When, on the other hand, the metal enters at several levels successively, the bottom of the casting is filled by the first metal entering the mould, and hot metal is run in on top of it through the upper gates; and in addition, the freezing of the lower portions is hastened by the chills. We thus get the full benefit of both the chills and the pouring in layers, so that as just stated their effects are cumulative.

In the proportioning both of gates and runners, and of sink heads and necks, the laws of fluid pressure are of great importance. Misconception of these laws frequently leads to the adoption of means of feeding and pouring castings that result in great waste of metal scrapped as sprues, without in any way increasing the tendency to make the casting sound.

Briefly stated, this law is that in any body of liquid, be it water or fluid steel, the pressure per square inch at any given point is

dependent only upon the depth of that point below the surface; and that the pressure at this point is equal in all directions. Thus, the several runners shown in Fig. 17 all exert the same pressure on the 1-in. gate—there is no more tendency to "force the metal into the mould" in the case of the large runners b, c and e, than in that of the small one a. That the pressure in each case is the same we



know, because if we join, for instance, a and b, as shown in Fig 18, and pour both full of liquid, the level of the surface of the liquid will be the same in both. Now if there were a greater tendency in b to "force the metal through the gate," obviously the metal would rise in a.

A second point of importance in this connection is that when the metal is at a higher level in a, for instance, than in b, it tends to flow into b; and the effective pressure on the metal in the gate is proportional to the difference between the level in a and that in b. As b fills up, this effective pressure decreases, and becomes zero when the metal is at the same level in the two sides.

If b is fuller than a the same condition exists, but the direction of flow will be reversed. The fact that b is of greater cross-section than a, however, does not make the effective pressure at the gate any greater when the level of the metal in b is, for instance, b in. above that of the metal in b, than it is when the level in b is b in. higher than that in b.

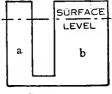


FIG. 18.

This brings us to the consideration of the proper relative proportion of the sizes of gate and runner. The latter should obviously be large enough to carry the metal into the mold as fast as it is needed; therefore, it must be able to carry away the proper sized stream of metal as fast as it is poured from the ladle. Now if this is the case, and the gate or gates are of smaller cross-section than

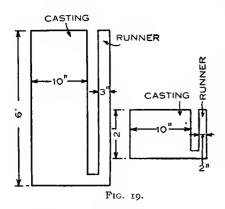
the runner, the latter will fill up with metal rapidly. The difference in level between the metal in the casting and that in the runner will then increase, putting more pressure upon the metal in the gate and increasing the velocity of its flow. This will be disadvantageous, first, because it will tend to cut out the sand of the mould, and second. because it will diminish the rate at which the casting as a whole is filled, and especially the rate at which the last metal enters the mold. For while the velocity of the entering stream will be increased as the runner fills up, so that at first the quantity of metal entering the mould will be maintained at nearly its proper amount; yet when the runner is full, the velocity cannot be further increased, and will as a matter of fact fall, as the casting fills and the difference in level between the metal in the casting and that in the runner decreases. As a result, the metal toward the last will enter the casting very slowly, and the gate and lower part of the runner may even freeze. We see, therefore, that the area of the gates should equal that of the runner.

To put this somewhat differently, and perhaps more exactly, the average rate of flow is determined by the size of the gate, and the average difference of level between the metal in runner and casting during the pouring of the mould. If the gate is so much smaller than the runner that the latter is full practically from the time pouring begins, the flow at first will be proportional to the height of the runner, and at the last will be zero, the average flow being proportional, then, to half the height of the runner. If the gate and the runner are of the same size and capable of carrying the metal as fast as it is run in, the difference in level between the metal in the casting and that in the gate will be practically uniform throughout; since this level will be that which suffices to overcome the friction of the flowing metal against the sides of casting, gate and runner. As the metal grows cold in the casting and its fluidity decreases, this friction will increase, so that the difference in level will gradually increase. Thus the flow will be fairly uniform throughout, and a considerable decrease in the rate of pouring toward the last will be voluntary (as by decreasing the rate of pouring), rather than involuntary.

Thus the cross-section of the gate or gates should be equal to that of the runner; in most cases there will be no object in making it greater. Should there be a number of gates, however, whose total cross-section is equal to that of the runner, they may be individually so small as to be in danger of freezing. In this event, their total

cross-section may be made greater than that of the runner. It should be remembered, too, that steel, especially low carbon steel, has so high a melting point that it is comparatively sluggish at temperatures that make cast iron as fluid as milk. The gates for steel castings are therefore made much larger than those used in pouring iron.

In filling a high casting through a single gate at the bottom, the metal that first enters the mould often becomes cold and semi-solid, and by its friction against the side of the mould greatly retards the rate at which the metal rises. The more slowly the mould is filled, the more pronounced this retardation. Hence, we must use a gate and runner of large cross-section; for if they are small the mould will be filled so slowly that the metal first poured will become very much chilled before it has reached its final resting place, this chilling will



greatly decrease the rate at which the metal flows into the casting, and the small gate and runner will be very apt to freeze shut. Hence a high mould requires larger gates and runners than a low mould of the same cross-section, resulting in a greater loss of metal scrapped as sprues per pound of casting in the case of the high mould. Take, for example, a single casting 10 in. in diameter and

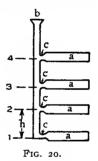
6 ft. high, requiring say a 3-in. runner, and three castings each 10 in. in diameter and 2 ft. high, each requiring only a 2-in. runner as shown in Fig. 19. We see that the weight of the three castings each 2 ft. high equals that of the single casting 6 ft. high. But the weight of the single 3-in. runner is greater than that of the three 2-in. runners in the ratio of 2.25:1. We scrap, therefore, about twice as much metal per pound of casting in making the single high casting as we do in casting the three low ones.

This brings us to the second advantage of the use of several gates at different levels in pouring a high casting, which was mentioned above. We see that if we use one runner and three gates 2 ft. apart vertically, in pouring the single 6-ft. casting, we can make the runner only 2 in. in diameter with perfect safety. For the lowest gate then

is required to feed the metal only so fast that it will flow freely until the second gate is reached; whereupon, the pouring practically takes a fresh start on the second section of the casting, the metal flowing in through the second gate. We no longer have to lift the sluggish metal that first entered the mould, and the metal in the runner below the second gate may freeze and welcome; the new gate will go on filling the casting. Thus in addition to improving the feeding by giving us partial top-pouring, the use of several gates leads to an actual saving of metal scrapped as sprues per pound of casting.

Multiple Pouring.—It appears at first glance that a similar saving in weight of sprue scrap is effected by "multiple pouring" of small castings, by which a number of flasks are stacked on top of each other and poured through a common runner. A little examination of the subject shows us that it is not, at least as far as the diameter of runner that we can use is concerned. Reference to Fig. 20 will make this plain. Thus, if a-a be a series of small castings poured through a common runner b, by means of gates c, we see that the pouring of the lowest casting and of the lowest section of the runner,

from level 1 up to level 2, is in no way affected by the fact that there is additional runner extending above level 2. The size of runner and gate necessary to pour the lowest casting is exactly the same as if this casting were being poured alone. The runner then fills up to level 2, and the second casting is filled just as if there were no others either below or above it. No reduction in size of runner can be made on the score of reduced friction of cold metal rising in the mould, because there is never any question of pouring castings



of greater height, but merely of a runner of greater height. Moreover, the height of the section of runner appertaining to each casting is not necessarily any less than the height of the runner used in pouring the castings singly.

There are, however, two reasons why multiple pouring effects a certain saving in runner scrap. The first applies to the cases in which the runner acts as a sink head and feeds the casting after pouring ceases. When the castings are poured singly, the runner has to be so thick and high that it will stay molten long enough to provide the necessary metal to feed the casting. With a common runner for several castings at different levels, the metal in each section of the runner helps to feed the section below, both during

and after the pouring of the lower section, and thus the length of runner per casting (and possibly the diameter) may be somewhat reduced.

The second source of saving is in the cup-shaped tops of the runners. If we use but one runner for six or eight castings instead of six or eight runners each with its small cup at the top, we save just so much metal. This saving, however, is not necessarily considerable, as the use of a cup-shaped top on the runners is not always essential. There will also be a certain saving in spatterings, since we shall fill up to the top of but one runner for say six or eight castings, instead of to the top of six or eight runners. It is by no means necessary to run metal over the top of the flasks in pouring, however, and a good gang will not lose much metal by slopping, so that this saving will not amount to much with careful workmen. Thus the chief advantage of multiple pouring is not so much the saving in sprue scrap, as the economies that it allows in floor space, top weights in the case of snap flasks, and pouring time.

Proportioning Sink Heads.—From what has already been said it is apparent that in the designing and locating of sink heads, it is important so to place the casting in the mould and so to distribute the chills that the maximum amount of feeding will be done (1) during pouring, by the hot metal being run into the mould; (2) by the heavy sections of the casting which feed the light sections placed below, and (3) by the heads during the time when they are actually being filled (preferably from the top, so that they will be poured with fresh hot metal). In many cases it is impossible to take advantage of all of these methods. By careful arrangement, however, it is generally possible to utilize one or more of them, so that the heads have the least possible amount of feeding to do after the pouring is stopped.

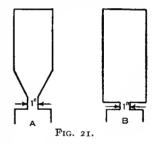
Thus in designing and placing heads we first consider how far we can take the work of feeding away from them and put it somewhere else. Then we consider the size of the head to be used, in view of the size of the section to be fed, and especially in view of the size of the neck through which the head is to feed.

In a great many cases, the thickness of the neck is fixed by that of the portion of the casting upon which the head is set. This being true, we are again confronted with the laws of fluid pressure which were mentioned in the discussion of gates and runners. That law teaches us that if the diameter of the neck is small, it is, from one point of view, quite useless to use a very wide head, because such a head exerts no more pressure upon the metal in the neck than

would a head of equal height and of the same diameter as the neck. As far as exerting pressure upon the neck goes, only the height of the head is effective, and from this point of view the best head is one of diameter equal to that of the neck and of such a height that it shall contain enough metal to do all the feeding that will be needed.

But there is, of course, another side to this question, which is that a high and very narrow head would inevitably freeze to the center and cease to feed long before the cavity in the casting had been filled up, or to speak more accurately, before the cavity had ceased to form. We must, therefore, make the head of such a diameter that it shall not freeze to the center until after the necessity for its remaining molten ceases to exist—in other words, the head must be at least as large, generally larger, than the section of casting it is to feed. It is quite useless, however, to provide a head of just

the right proportions, if the neck freezes before the head has ceased to feed. Hence, the diameter of the neck must be great enough to prevent its freezing prematurely. The rate of cooling of a neck of a given diameter, moreover, is greatly influenced by its length and by the shape of the head. Thus a long neck, tapering upward to the full size of the head, A, Fig. 21, will cool much

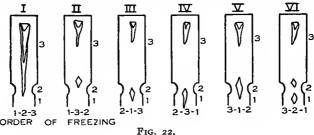


more rapidly than a short one of the same thickness, as shown at B in Fig. 21, because in the latter the short neck is kept hot by the closely adjacent mass of metal in the head and casting. The neck should, therefore, be made as short and be joined to the head as squarely as it is possible to mould them.

In many cases, of course, the sink-head is not necked in at all, but is brought down to the casting at its full diameter. When the section to be fed is large enough to permit the use of such a head, the danger of premature freezing of the neck is eliminated.

We see, therefore, that when the diameter of the neck is fixed, as for instance by the width of the portion of the casting to which it is attached, the neck should be made as short as possible, and the head should be made with a square bottom and no wider than will suffice to keep its interior liquid and maintain the feed up to the moment when the neck freezes. All extra width beyond that necessary to accomplish this is so much plain waste of metal. The height of the head also should not be greater, for a given diameter of

head and neck, than is necessary to provide enough metal to feed the casting completely. The head that has done its work to perfection would be hollow just down to the neck. Frequently we see heads 8, 10 and even 12 in. in diameter and 18 in. or 2 ft. high, with necks only 2 in. wide and 3 or 4 in. high. Manifestly, such heads have contained fluid metal and been capable of feeding the casting long after the neck, and even the casting itself, have solidified.



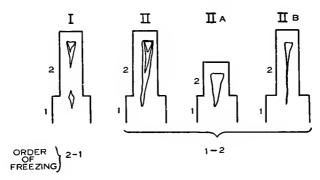


FIG. 23.

The reservoir of liquid metal in the head is quite useless for the purpose for which it was provided, like a power dam with the flume to the turbine choked with mud—the liquid we require to do our work is there, but it cannot flow to the place where we wish to use it.

The factors governing the proper size of head and neck for a given casting vary to such an extent that it is practically impossible to do more than estimate roughly, by the light of previous experience, how large they should be made, and then allow an ample margin for safety. Having made a casting with a given size of head, it may prove unsound, that is, it may show a pipe under the neck

when the head has been knocked, or cut from the casting. In that case, the best means of remedying the trouble has to be considered. Again, the casting below the neck may be sound. Generally, it is assumed in that case that the proportioning of the head has been correct, and so it has, as far as making a sound casting is concerned. Often, however, the margin of metal that has been provided is too great, so that the waste in sink head scrap is excessive, and it will pay to consider how much the sink head may be reduced in size without risk of making an unsound casting.

If the cavity is deep down in the casting and separated from the sink head and neck by a layer of sound metal, the trouble obviously is that the upper section of the casting did not remain fluid long enough to feed the lower section—the error is in the location of the head, or the position of the casting in the mould. The casting must be poured so that the last freezing portion is above, not below, the first freezing portion. In most cases such a cavity will not be detected unless the casting is cut open for inspection, or breaks in service.

Theoretically, given a section of casting to be fed, a sink head, and a neck connecting them, there are six different orders in which the three may freeze. These are represented diagrammatically in Fig. 22, numbered cases I to VI, with the order of freezing indicated and the shape and location that the cavities may be expected to assume sketched in. Of these cases II, V and VI are of little interest, as they all involve the freezing of the head before that of the neck; a condition that will seldom be found to occur in practice, at least under ordinary circumstances.

Taking first case VI, we see that as the head, neck and casting successively freeze, the metal at the bottom of each being sustained by the still liquid metal beneath, each will form its own pipe; two of which will probably be more or less double-ended, as shown in the figure.

Similarly in case V, the head will form its own pipe, then the casting in freezing and piping will in all probability drain the neck, which will then freeze last and pipe at the top, forming a continuation of the pipe in the casting. Should the neck contain enough metal to feed the casting completely, this pipe of course would be only in the neck and not extend into the casting.

In case II, the casting would freeze first, and its pipe would be filled by the liquid metal above. Then the head would freeze and pipe, and lastly the neck would freeze and pipe.

Should the head freeze before the neck, as in cases V and VI, it would generally be found that the shrinkage of the casting that took place before the head froze would drain the neck, so that in either case there would probably be a more or less continuous pipe running through the neck and up into the head. The same would often be true in case II; so that a neck which has frozen later than the head will generally be hollow, the pipe going a little way into the head and in many cases extending into the casting. The remedy is a wider head, which in some cases may be made shorter; this being determined by the depth of the pipe in the upper part of the head.

In practice, cases I, III and IV will occur. Case I may involve a head, neck and casting all piped, as indicated by the deepest pipe shown in the sketch. In that event, there was obviously not enough metal in the head to feed the casting. The remedy depends upon the thickness of the shell of the frozen head and neck. If it is very thick, and if the pipe is but a very narrow hole, it is plain that if we secure more metal by increasing only the height of the head, the head and neck will freeze tight before this extra metal can reach the casting, and the situation will be similar to that shown in case III. We need, therefore, a wider head and neck, and perhaps a higher head also; since increased height gives greater pressure on the neck, and will force more metal through it when it is frozen nearly to the center.

If the walls are very thin, showing that the metal ran out much faster than it froze, increase of the height of the head, and a slight increase of the diameter, are called for. This case, however, may also indicate that if we had kept the ladle over the head a little longer and fed the shrinkage by slow pouring at the last, we should have come out all right with the head and neck we had.

On the other hand, if the casting and neck are sound, and the pipe in the head comparatively shallow, we have overdone the matter and wasted metal in the head. The latter may be reduced in height and probably also in diameter, until we find it is piped nearly to the bottom, as shown in the intermediate positions of the pipe in the sketch.

Cases III and IV are those in which the neck froze first, by far the commonest cause of piping in castings. In case III the final solidification of the casting occurred next after that of the neck, while in case IV the casting froze last. The chances are that the pipe will be deeper in case IV, because the beginning of solidification and pipe-forming in that case came longer after the neck closed, so that less

feeding in the preliminary period of pipe-forming occurred. This pipe will run up into, but not through, the neck; since the upper part of the neck will have been fed by the head in both cases. There will in both cases be a pipe also in the upper part of the head.

The remedy is first and foremost a wider and if possible shorter neck, as it is not a question of not enough metal to feed, but of the feed channel being prematurely choked. The size and depth of the pipe in the upper part of the head will indicate further whether the head that was used contained too much or too little metal to complete the feeding had a neck of the proper size been provided.

In case a narrow neck is unavoidable, it should be kept in mind that the narrowness of the opening decreases the flow of metal from the head, especially as the opening grows smaller, by the friction of the stream of metal on its walls. It may well be, therefore, that the opening in the casting grows too rapidly to be filled by the metal from the head during the last part of the time that the latter is still actually feeding. More pressure on the metal in the neck will increase the flow, and this pressure is obtained by increased height, not increased diameter, of head.

It will be seen from the above discussion that to make the necks small, in order that the heads may be knocked off by means of a sledge hammer or drop ball or burned off at the least expense, is dangerous practice because it may well lead to premature freezing of the neck. The result will be a pipe in the casting which will have to be plugged by means of the welder, the expense of the job often more than balancing the saving made in the cost of removing the head. Moreover, when the pipe is irregular and partly discontinuous the plugging often merely fills the top of the opening, like a cork in a bottle, and leaves the casting dangerously weak. Many a broken casting shows this defect in a striking manner. We may get our work accepted on a first order by indiscriminate plugging of holes, but if many of our castings break and show unsound places we will be pretty sure to find the customer has gone elsewhere for further orders.

An extreme case of such bad practice is illustrated in Fig. 24, taken from an advertisement published a few years ago by the makers of a certain cutting torch. The casting shown weighed 200 lb., and had six risers varying from $2\frac{1}{2}$ by 7 in. to $4\frac{1}{2}$ by 15 in. in section, which were cut off, according to the advertisement, in 20 minutes. It is plain that the necks of these risers are so small that the greater part of the metal put into the heads was simply wasted; for all the good

it did in feeding the casting, it had as well been poured on the ground. In this particular case the necks could not be made much wider, and heads one-third the size used could have been cut off quite as rapidly, and would have fed the casting as well.

By moulding the neck in a special core it can be made very short, so that although considerably narrower in at least one dimension than

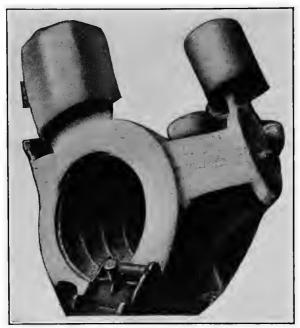


FIG. 24.

the head, it will feed very efficiently. The core is so thin and the neck consequently so short, in this method of moulding, that the mass of hot metal in head and casting quickly brings the core to a high temperature and thus keeps the neck fluid. The result is a very perfect feed with a neck comparatively narrow in one dimension. Moreover, by properly shaping the opening in the core the neck may be so formed that it can safely be broken from the casting, even though quite heavy. Care has to be exercised, however, not to knock off too heavy a neck or one that is not shaped just right (caused in some cases by the partial disintegration of the core), as the heavy shock of the blow brought upon a casting already full of shrinkage stresses may crack the casting. This question is con-

sidered in more detail in the discussion of "hot" and "cold" cracks in castings.

When the head is the same size all the way down and no neck is used, there are but two orders in which freezing may occur, head first, or casting first. These cases, and the probable shape of the resulting pipe, are shown in Fig. 23, page 254.

When the head freezes first, it will form its own pipe; the casting will then freeze and the pipe formed in it will probably run up a little way into the head (case I). When the casting freezes first, it may be sound (case II). Then if the pipe in the head reaches down only a little way, the head has been made too high and perhaps too wide. If the pipe in the head goes quite deep but does not reach the casting, the size of the head is theoretically correct. Should the pipe reach into the casting, it may be very wide, so that the walls of the head are thin (case II-A); this calls for a higher, and perhaps a somewhat wider head. If the walls of the head are very thick and the bottom part of the pipe a small hole, greater diameter of head is the obvious remedy (case II-B).

It is, of course, impossible to give hard and fast rules for such reasoning, especially in view of the fact that the shrinkage of the metal in the plastic state that succeeds solidification changes the shape and size of the interior cavities. Their location and shape may be such that exact deductions from them are impossible. It will, however, be advisable in a great many cases to consider the remedies to be applied along the lines suggested above, rather than to jump blindly from the obvious fact that a shrinkage cavity has occurred, to the conclusion that a wider head is the remedy.

It is plain that as the metal in a head flows, or tends to flow, by gravity, the action of the head is most efficient downward. The extent to which a head will feed an opening that extends chiefly in a horizontal direction is evidently not great. Hence, when a considerable extent of horizontal surface is to be provided with heads, especially if the vertical dimensions of the castings are small, a number of heads must be used and they must be placed so close together that their effects overlap. Such a casting will be more efficiently fed by a number of comparatively small heads placed close together than by a few heavy heads.

Sometimes, too, efforts to cast a shape "on the flat" by the use of heads attached to the broad side of the section prove utterly futile—and then some fellow comes along and casts the thing "on end," and secures a solid casting with a sink head of half the size and weight

used before. As sink heads are so much metal on which money is spent in melting, without return as salable castings, it is needless to say that the lighter the sink head the more money the shop will make.

The principle just stated, that a sink head acts best downward, is of course, simply another way of expressing the obvious fact that when a casting is poured "on end," each section of the casting, from below upward, feeds the section immediately under it and the head feeds the upper sections. A further precaution suggested by this line of thought, and one that has been mentioned already in discussing the size of sink heads, is so to locate the casting in the mould that as far as possible the upper portions will freeze last, and thus feed the lower portions and be fed in turn by the heads.

An error sometimes committed is to use heavy heads with thick necks, attached to the sides of castings near the bottom. The heads are made very heavy, evidently with the idea that their total weight can be utilized in forcing the metal they contain up into the casting to feed the shrinkage of the latter. If they fail to feed as expected, heavier, but seldom higher ones, are tried. Of course, to apply again our laws of fluid pressure the effective pressure exerted upon the neck of the casting shown in Fig. 25 by the metal in the heads, is proportional only to the difference in level A-B between the metal in the head and that in the casting, so that after pouring is stopped only the portion A-B of the head is effective in producing pressure on the metal in the neck to make the latter flow. Increased diameter of head will not increase this pressure in the slightest though by keeping the metal hot longer, it prolongs the period during which the pressure is exerted. Increased height of head, on the other hand, greatly increases the pressure.

Moreover, the heads, so attached are laboring under the disadvantage that they do not feed their fluid metal directly downward into the top of the gradually deepening pipe, but have to do their work by bringing pressure to bear on the still liquid metal in the casting, by which that metal is made to flow upward and close the pipe as it does when a solidifying ingot is compressed at the bottom. Working in this manner, the heads must fail to be effective as soon as the metal in the casting becomes too sticky to flow and they cannot, however long they stay fluid, feed the portion of the pipe that forms after the metal of the casting has become solid. A head placed on top of the casting, on the other hand, continues to feed the pipe so long as the head and neck remain fluid, and it feeds chiefly by

direct flow, not by forcing the metal in the casting to move upward. Hence, it feeds efficiently at all stages of the formation of the pipe. Heads placed in the manner shown in Fig. 25 must inevitably be seriously inefficient, and in a great many cases might as well be omitted altogether as so attached.

In the case of comparatively thin pieces, however, where the sections are fairly uniform and the horizontal area of the casting considerable compared to its thickness, heads attached to the sides in this manner sometimes feed better than heads placed on top of the casting. On cutting off heads placed on the top, spongy places are often found under them, whereas when the heads are attached at the side by means of a heavy neck, the casting will be found

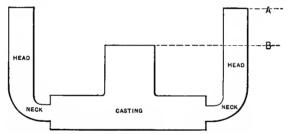


Fig. 25.—Ineffective manner of attaching sink-heads.

perfectly solid. The reason for this appears to be that the shrinkage of the casting when headed and gated on the side takes place largely while it is being poured, and the casting is fed chiefly by the metal being run in through the gate. The heads, one of which is often used as a gate, are filled with metal of about the same temperature as that in the casting, and are called upon to feed only such shrinkage as occurs after the mould is full. As the heads are very much higher than the casting, they are called upon to feed metal only horizontally, not horizontally and upward into the casting. When, on the other hand, heavy heads are attached to the broad face of the casting, the metal that rises into them has been in the mould longer, and is consequently colder, than the metal poured in last and forming the body of the casting. The shrinkage of this hot metal, apparently, sometimes occurs after the head has grown too cold to feed properly, with the result that the metal under the head is more or less unsound.

Tipping up Shallow Castings.—Thin castings, especially long, narrow, shallow sections are frequently tipped up at the gate end for

pouring. The most obvious reason for doing this is that it assists the pouring by making the metal run down-hill, instead of along on the level, and so enables us to "run" a piece that we might not be able to fill in any other way. A further advantage of this method, however, is that it to some extent substitutes top pouring, and downward feeding by the heads along the length of the casting, for bottom pouring. The more we tip the casting up, the greater this advantage; indeed, whenever possible, we pour the thing "on end," so that when it is filled each section may feed that immediately below it, and a comparatively light head on top will suffice to give us a sound casting; while if we poured "on the flat" we might need several heads distributed along the casting. In some cases such a casting can be made sound by the action of chills placed upon the lower end, without the use of sink heads. The metal running in through the gate does much of the feeding while the casting is being poured, and the metal in the runner suffices to feed the final shrinkage that takes place after pouring ceases.

The sink heads serve to carry off the loose sand that has been torn from the sprue or the casting by the flowing metal, or that has spalled off from the mould as the metal rose; all this "dirt" floats on the steel as it rises in the mould, and will be trapped against the cope unless sink heads are so placed and so shaped that all of the top metal enters them. Small heads, covering only a part of the top of a casting cannot be expected to carry off all this dirty steel, and will not do so. Thus in the case of cylindrical castings poured on end, the head should be a continuation of the casting, a so-called "annular" head; one or more small heads at intervals around the top will not serve. To reduce the weight of so large a head it may often be given a tapered cross-section. Indeed, even with plain heads, a great deal of metal can often be saved by giving the heads the shape of a frustrum of a cone, rather than a cylindrical shape; the large end of the cone, of course, should be down and the small end up. This brings the greatest mass of metal next to the casting and neck, where it can serve most effectively to keep the neck from freezing prematurely, so that of a given amount of metal in the head a larger proportion can be drained out than when the same amount of metal is used in a cylindrical head. This practice, however, should not be carried so far as to decrease the amount of metal in the head beyond the minimum necessary to properly feed the

Castings which have to be poured "on the flat," using only a

limited number of heads, are particularly difficult to cast free from sand and slag pockets on the cope side. Sometimes it is desirable to pour such a casting in an inclined position, with the gate down-hill and as many of the heads as possible near the end tipped up; in this way the dirt can often be carried up through the casting and washed into the heads at the highest point of the casting as it is placed for pouring.

Cracks.—Cracks in castings are due primarily to the shrinkage of the steel in cooling from the temperature of solidification to that of its surroundings. The shrinkage of ordinary carbon steel amounts to about $\frac{1}{4}$ in. per foot, being somewhat less as the carbon approaches .00 per cent., and greater the lower the carbon. Could all parts of the casting cool at the same rate, shrinkage would make less trouble than it does. But when a light portion cools ahead of a heavy one, the resultant shrinkage of the light section often brings a pull on the heavy one when the latter is still hot and weak, resulting in the tearing apart of the two at the weakest place—which is not necessarily the smallest section, but that which is weakest at the temperature existing when the stress is set up. Should the casting hold together, the later shrinkage of the heavy portion after the light part has become cold and rigid often sets up heavy compression stresses in the latter. Thus in the shrinkage stress test bar shown in Fig. 26, the light section cools first, the hotter heavy part yielding to the pressure. Then when the heavy portion begins to cool faster than the light part, the latter is compressed very strongly, so much so that if it is cut in two with a hack saw when cold, the moving together of the sides of the cut can be measured very easily.

Cracks are favored by:

- 1. High sulphur in the steel, causing red-shortness.
- 2. Low carbon, which gives the maximum shrinkage.
- 3. Unevenness of sections, causing variations in the rate of cooling.
- 4. Sharp corners in angles of castings, especially where a thick section joins a thin one.
 - 5. Use of tie-bars stronger than the casting.
 - 6. Resistance of mould and cores to shrinkage of the casting.
 - 7. Pouring very "hot."

We are obliged to make our castings either of a definitely specified carbon content, or of a low enough content to make the annealed castings tough and easily machined. We cannot, therefore, alter the carbon content to decrease the shrinkage. The sulphur content depends upon the raw material and the steel making process

we use. Basic open-hearth steel can be kept quite low in sulphur, basic electric furnace steel even more so, without very great expense. Acid open-hearth, Bessemer and acid electric furnace steel frequently run rather high in sulphur, so that the foundryman who uses them is obliged to exercise more care in guarding against cracks than the user of basic steel. The effect of high sulphur content is to make the steel weak and brittle when hot (red-short), so that it more readily tears apart under shrinkage stresses.

The means by which the foundryman seeks to prevent cracks are:

- r. The use of chills.
- 2. The use of fillets.
- 3. The use of brackets.
- 4. The use of tie-bars weak enough to break before the casting does.
- 5. Breaking up the mould and core.
- 6. Cooling castings very slowly.

Thus the second function of chills is to prevent hot cracks in the castings. In a way this is identical with their first function, because in both cases they work by hastening the rate of cooling of one part of a casting as compared to that of another part. But it is in fact a distinct function, in that the ultimate object sought is not to make the casting sound, but to prevent the section which normally cools the faster from pulling away from that which normally cools the slower. A single chill may in some cases exercise both functions.

As we have seen, a light section that cools ahead of a heavier one to which it is integrally attached, often pulls away from the heavier portion so strongly as to tear the metal apart where the two join. Hence, we place chills upon the heavier part, that it may cool and shrink as fast as the lighter portion; and especially we chill the junction point, where the failure tends to come, that it may be sufficiently cold and hence strong when the pull comes to endure the stress without failure. If the steel holds we have attained our first object, but we should not forget that the stress still exists and that the casting must be handled tenderly until the stresses are relieved.

Fillets are so well understood that an extended discussion of them is not needed. They consist simply in a rounding off of the sharp interior angle corners of a casting, especially where thick and thin sections join, by means of suitable design of the pattern. The result is that no sharp corner is presented as a starting place for a crack.

Brackets serve the same purpose, and need no more discussion than do fillets. They consist of extra metal in the shape of tie pieces (generally triangular) across corners and places which are apt to crack, and may be considered as portions of a discontinuous fillet of large radius. Both fillets and brackets serve chiefly to reinforce the corners where failure is likely to occur.

Tie-bars are extra pieces of metal used to bridge across from one part of a casting to another, and serve to prevent the shrinkage from warping the casting out of shape. When they are made too strong they may serve their true purpose so well as to overdo it. For if the shrinkage stresses which they resist should become so great that something must go, obviously the weakest part will be the victim. Hence, the tie-bar often has to be so designed that it will be sure to fail before the other portions of the casting; and in some cases it is notched to insure that it and not the casting itself shall give way under the stress, should the latter be great enough to break something.

Many castings are so shaped that the resistance of cores or portions of the mould to the shrinkage of the steel will result in the cracking of the work unless the sand is thoroughly loosened up as soon as the steel is solid. The fillets and brackets, chills, etc., may prevent cracking at the time of solidification, but further cooling with the shrinkage that accompanies it will crack the casting unless the enclosed sand is so loosened that it does not oppose shrinkage. This can be taken care of to a certain extent by making cores in such a manner that they will crush easily when the shrinkage of the steel brings pressure to bear upon them, and (especially in the case of green sand moulds) by not ramming the sand too tight, so that it will yield to the pressure of the contracting casting. But many cores, or portions of the mould proper, which are so located that they will seriously oppose the shrinkage of the casting, must be broken up with bars soon after the metal solidifies. Frequently also, the resistance of the sand in the cope to the motion of the sink-heads as the casting cools will have the same effect unless the cope is broken up.

In many cases of large and especially of long castings, whose total shrinkage is naturally great, special arrangements are often made in the moulding to facilitate breaking up the mould. The most striking case known to the author was in published descriptions of the practice followed in making large ship castings, when by means of a properly arranged set of pipes, the sand of a large portion

of the drag part of the mould was washed out with a powerful stream of water.

Another result of this resistance of the mould to the shrinkage of the steel is to alter the dimensions of the piece to an extent depending upon the total shrinkage and the degree to which it is prevented by the sand. Thus very long castings with many heads will vary somewhat noticeably in length depending upon whether or not the mould is thoroughly loosened up. The reason for this is, of course, that if the shrinkage of the casting is opposed to a sufficient extent to overcome its elastic limit at the temperature existing at the time, the steel yields to the stress and assumes a dimension not proper to it at that temperature. On further cooling, the casting does not shrink to the dimensions it would have attained had its shrinkage at higher temperatures been unopposed. This virtual expansion, which is analogous to that of the outside layers of an ingot which gives rise to pipes, is often sufficient in amount to make a serious difference in the dimensions of long castings.

The hotter the steel is poured, the greater the difference in the rate of cooling of thick and thin sections, and hence the greater the tendency of complicated castings to pull apart in cooling in the moulds. As the steel is probably of better quality when pounded as "cold" as it can be, and still run the castings, very hot steel should be avoided as far as possible.

Of miscellaneous sources of unsoundness and weakness may be mentioned the use of nails or chaplets to hold a core in a light section, which are too heavy to be melted by the small amount of metal poured around them and remain imbedded in the steel but not welded to it. The author has seen failures of certain light castings, every one of which had broken at a point weakened by the presence of nails that had not been melted by the metal surrounding them.

Though it is not exactly a case of unsoundness, the spoiling of a casting by the lifting of the cope may properly be discussed here, as it is due generally to misapprehension of the laws of fluid pressure. Let us take, for example, a circular casting having an exposed horizontal surface of 2 sq. ft., and provided with three heads each 1 ft. in diameter and 2 ft. high; and a similar casting having an exposed horizontal surface of 6 sq. ft., and provided with one head 4 in. in diameter and 2 ft. high. In both cases it is assumed that the casting is moulded entirely in the drag, and the heads in the cope. Now the upward pressures tending to lift the copes of the two castings when they are poured full of steel, are determined not

by the size of the heads, but solely by the surface areas of the castings and the height of the heads. Thus the pressure is three times as great in the case of the casting with one small head as in that of the casting with three heavy heads, since the height of the heads is the same in the two cases and the areas of the surfaces are as 3:1. Were the narrow head of the second casting twice as high as the wide heads of the first, the pressures would be as 6:1. This is easily understood when we remember that according to the laws of fluid pressure the pressure per square inch at any given point in a body of liquid is equal in all directions, and is dependent only upon the depth of that point below the surface of the liquid.

We thus are obliged to hold moulds together with very strong clamps, or to put very heavy weights upon the copes, in cases where though the total weight of the heads is small, their height and the surface area of the casting are considerable. The weights put upon a cope may thus have to be several-fold heavier than the metal in the heads.

Pouring.—To secure the best results in running castings, the rate of flow of the steel should be under control. That this is true will be at once apparent if we consider the extreme, and of course impossible, case of a mould filled instantaneously. The metal being then at an even temperature throughout, cooling will begin at all parts of the surface at once, thin sections will cool much before thick ones. setting up heavy stresses; and the entire work of feeding the shrinkage will fall upon the sink heads. In some cases very rapid pouring, approximating these conditions, does not reduce the effectiveness of the sink heads—but frequently, the result of rapid pouring is that the top of the casting cools as fast as the bottom, so that the feeding is very incomplete. Generally it is most desirable to get the first metal into the mould as rapidly as possible without cutting into the sand, and gradually decrease the rapidity of flow as the mould fills. Such pouring, especially when the gating has been so arranged that the metal enters the mould at successively higher levels, or from above, allows the maximum proportion of the shrinkage of the solidifying bottom portions of the casting to be taken care of by the hot metal being run in, and leaves the sink heads less to do when the pouring is stopped. The heads when possible should be filled from above with hot metal.

The desirability of being able to control so nicely the rate at which the mould is filled is the chief argument against the use of ladles that pour through the bottom with stopper and nozzle in the usual manner. The bottom pour ladle gives a stream that runs with great velocity, and the velocity is dependent upon the depth of metal in the ladle. To check the flow of steel, we can only lower the stopper head, which results in a fan-shaped stream that strikes the pouring cup instead of running cleanly down the runner, and the wear on stopper heads prevents us from indulging in this practice too freely, lest we cut the head off.

Large ladles, of course, cannot be poured with nicety over the lip, but can be handled only with a nozzle. Used for castings of such size that the full flow of the nozzle is required for some minutes (and in very large work it is often hard to get the steel in fast enough), the nozzle does excellent work. But for small castings it is generally best to use a ladle of not over 3 tons capacity and pour over the lip. It is even probable that many of the moulds that are commonly poured from a 10-ton ladle with nozzle, could be far better handled by two or three light ladles filled from the large one—the big ladle meanwhile handling the heavy work. To follow this practice makes necessary some means of tipping the big ladle to pour into the small ones, which is not commonly provided.

Again, when we have to pour very light castings, the 2- or 3-ton ladle is too big, and its stream is too heavy and falls too far. would take too long also to pour a great many small flasks from the big ladle. "Bull ladles" or "shanks" are needed for this work, and are used in sizes of from as light as 50 lb. capacity up to 1000 lb. or so, depending upon the size of the castings to be poured. For a given weight of castings, say an average of 5 lb. each, it is questionable whether the very small shanks sometimes used are desirable. Some foundries making 2-ton heats use 100-lb, shanks, pouring 20 castings from each, working five shanks at once; and thus are obliged to refill each shank eight times. The metal to be sure is held in the shanks a very short time, and hence has little time to cool off; but in such small masses it cools rapidly, and the time lost in running back to the big ladle to refill increases the time spent in pouring the heat, and hence the cooling off of the steel in the big ladle. To save the time spent in refilling ladles it is desirable to use the largest shanks that can be poured without allowing the steel to grow "cold" in them.

When heats of some 2 to 3 tons are made, as in the majority of the small Bessemer shops, many heats are poured partly from the large ladle, party from the shank. In this case, the first of the heat is, of course, used for the light work, poured from shanks, and the pour-

ing is kept up until the metal becomes too "dull" to run the light castings, when what remains in the ladle is used for heavier work. The advantage of providing both light and heavy work for each day's pour is too plain to require extended exposition, as the heavier work uses the dull metal from the last of many a heat, which would have to be poured as scrap if only light castings were made up for pouring.

In small Bessemer foundries (and occasionally in small electric furnace work), the relative advantages of "shanking from the vessel" and "shanking from a ladle," are in debate.

The advocates of the former method base their arguments upon the fact that the hot vessel keeps the steel up to temperature better than the ladle. On the other hand, the advocates of pouring the whole heat into a ladle point out that filling shanks from the vessel is slow and rather dangerous, so that the heat is poured more rapidly by shanking from a ladle—and further that by taking the steel at once from the vessel and starting another heat, the vessel is kept very hot, with the result that the steel of each heat is hotter when blown. The advantage probably lies with shanking from the vessel when several rather large shanks are used, and access to the vessel for filling is made easy, and with shanking from the ladle when light shanks are employed, and access to the vessel is difficult.

The foundry lay-out governs to a very great extent the pouring methods that can be followed. This will vary so widely from shop to shop that only general consideration of the subject is possible. The lay-out for the day's pouring should be so arranged that a definite routine can be followed with each heat. Care should be taken that the moulds are not so distributed as to necessitate carrying the ladle long distances in moving from casting to casting. Should several ladles be necessary, especially if each ladle is serving several shanks, the floor space must be ample, to avoid interference between crews. A little attention given to this subject in planning the shop when first built will save countless vexation delays when the productive capacity of the shop is reached.

Digging Out and Cleaning.—Frequently, as has been said, especially with steels whose shrinkage is high, a casting cannot be kept from cracking, unless cores or portions of the mould are broken up as soon as the steel has solidified. Large cores and portions of moulds are frequently made partly collapsible, for instance by making their interior portions of cinders or sawdust, so that they will yield to the pressure put upon them when the casting begins to con-

tract; but it is generally necessary also to break up the cores thoroughly. By making their interior of loose material, and the arbors of very brittle cast iron, the destruction of the cores with bars and sledges is greatly facilitated. Many castings, especially of some of the alloy steels, must be dug out of the mould and the cores thoroughly broken up the moment the steel is strong enough to bear handling.

Castings of very uneven section, especially of steel of high shrinkage, even if thoroughly dug out when red hot, will crack if allowed to cool in the air. The author has seen a 6-ton casting of ordinary carbon steel break clean in two through perfectly sound metal, some 36 hours after it was cast and some hours after it was cold enough to lay the hand on, from the strains set up by the unequal cooling of the thick and thin sections. To avoid this difficulty, the castings may be buried in sand and allowed to cool; or cleaned very roughly, charged into a furnace or heated pit, which is brought to about the temperature of the castings, and there allowed to grow cold. These precautions, in the majority of cases, will be sufficient to insure against cracking and breakage.

Many complicated castings of steel that has a high shrinkage like manganese steel, however, must not be allowed to grow actually cold before charging into the heat treatment furnaces; unless kept quite warm, they will break in cleaning, or in the first stages of heating up in the furnaces. Castings of this class almost always have to be taken from the sand and cooled in a preheated furnace or pit, mere burying in sand not being a sufficient safeguard against cracking, and are removed from the cooling furnace while still hot enough to boil water that falls upon them, when the heads and runners are knocked off, the sand and nails cleaned off as rapidly as possible, and the cored holes cleaned out. The castings are then got into the furnaces as soon as possible. Heads that cannot be knocked off without danger of cracking the casting are burned off with the gas torch.

Fortunately, these steels often have a low melting point and are very fluid, hence a neck very small in proportion to the sink head can be counted upon to feed the casting so thoroughly as in many cases to completely empty the head, leaving it a mere shell. The small necks, especially if the steel is brittle, make it possible in the majority of cases to knock off the heads, either by hand or with a dolly or drop, without fear of cracking the casting. Very good judgment, however, is needed in proportioning the necks and heads, in

placing the heads, and in knowing how far to go in hammering on a head that refuses to come off. Many a casting is destroyed by an overenthusiastic man who persists in pounding away at an obstinate head or runner.

Steel castings that can be allowed to grow completely cold before cleaning and annealing, either by merely shaking them out and leaving them to cool in the air, or by cooling them in a furnace, can be cleaned at leisure. In removing runners and heads, good judgment is necessary. They may be broken from the castings, when the necks and gates are small enough. It is a pretty safe rule, however. that if a head cannot be broken off by hand, or by a very light "dolly" or drop, it had better be burned or sawed, since it is apparent that a cold casting already full of heavy shrinkage stresses will almost certainly crack if these stresses are increased by rough handling. If the casting is of brittle steel, the head breaks off quite easily—but on the other hand the casting itself will break quite easily, too, under a suddenly applied load. If the steel is tougher, there is less danger of cracking the casting with a blow of given intensitybut the head will be harder to break off, so that the blow must be more severe. By cutting away the neck with an air hammer and chisel or the torch, many heads can be broken by hand; but to make the necks small enough to break, at the cost of bad feeding of the casting and holes under the head that require a lot of plugging, is poor economy.

The large heads, if accessible, can be cut off with a cold saw, and it is part of the business of the moulders so to place such heads that they shall be accessible, unless it is manifestly impossible to do so. In that case the oxyacetylene or other burning torch comes into play and either cuts the head entirely off, or so cuts into the neck that it can be broken without jar. For all but the very largest heads, indeed, the cutting torch is now almost universally used, and practically all foundries which are able to secure or make a supply of suitable gas have adopted it in place of the cold saw. The heads and gates are commonly removed from light castings with a "sprue cutter," a machine that does the work quite efficiently.

The sand, of course, is knocked off roughly with hammers or bars, after which the sand blast is employed to clean the castings thoroughly. Fins or small lumps are usually trimmed off with grinding wheels, air hammers or chisels, and any sand holes, pits and scabs are cleaned by the same means preparatory to welding.

For ordinary steel castings these operations, and indeed welding of

defects as well, are ordinarily completed before the castings are annealed. In the case of manganese steel castings, however, it is quite common practice only to rough clean before heat-treating the castings; the sand and nails are cleaned off with bars, heads and gates removed, fins trimmed with sledge or gas-torch, and cored holes cleaned out. The heating up and quenching that follow remove much of the sand that sticks to the castings too tightly to be removed with bars, so that as a rule sand blasting is needed only for a small proportion of the work.

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CHAPTER IX

HEAT TREATMENT AND ANNEALING

Even to-day, some foundrymen will assert that to anneal steel castings is superfluous, on the ground that the physical properties of low-carbon steel in the cast state are good enough. Ouite aside from the question of the shrinkage stresses remaining in unannealed cast steel, especially in large or complicated castings, the necessity of annealing should be beyond dispute; yet perhaps 50 per cent. (some authorities say 70 per cent.) of the steel castings made in this country are shipped unannealed. There are, of course, cases where weight is of no consequence, so that sufficient strength can be secured in the piece by increasing the sections, as is done with large cast-iron work; but when strong, tough steel is needed, annealing is absolutely necessary. To back up the contention that the untreated cast steel is strong and tough enough for most uses, the physical properties as shown by the usual static tests are said to be so-and so. "plenty good enough," say the advocates of unannealed cast steel. Far better static properties can readily be secured by annealing; but the main case for annealed steel rests upon the brittleness of untreated cast steel under sudden heavy shock, and upon its necessarily poor resistance to repeated stresses. Further on in this chapter will be found tests of a .10 per cent, cast steel which show that in the untreated state even so soft a steel as this breaks under heavy shock almost without absorption of energy, and without bending a measurable amount, though the static test bends through an angle of 50 degrees before breaking. Tests of unannealed cast steel under repeated stress, especially repeated impacts, will show to the satisfaction of any engineer that for service where real toughness is vital we cannot afford to dispense with annealing. merely to save a fraction of a cent a pound on a job. The author is convinced that if any intelligent foundryman should install equipment for testing his steel under repeated stresses and under impact, and study the data obtained by 6 month's careful testing. he would be the first to decry the practice of shipping unannealed castings.

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When the sections of a casting vary considerably in size, shrinkage stresses are set up in cooling in the moulds, due to inequalities in the rate of cooling of the different parts. The impression prevails very generally that the relief of these stresses is the chief function of annealing. Though this is far from the truth, yet it is an important function.

The magnitude of the stresses set up in the shrinkage of steel can be roughly measured by means of a test bar of the shape shown in Fig. 26. By measuring the distance between two punch marks, located as shown, on the thin portion of this casting, before and after cutting through this section with a hack saw, it will be found that this

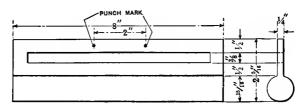


Fig. 26.—Stress-test casting.

distance after sawing in two will be about .or3 in. less than before. This amount, however, is the sum of the shortening of the large section, and the lengthening of that portion of the small section lying outside the punch marks. In this case the punch marks were 2 in. apart, and the length of the casting 8 in. Therefore this sum of .or3 in. equals the extension of three-fourths of the small section, plus the contraction of the large section. If we call the total extension of the small section X, then the contraction of the large section equals .or3 $-\frac{3X}{4}$.

Now the stresses in each section in pounds are equal. The cross-sectional areas of the sections are .6902 sq. in. and .125 sq. in.

If the modulus of elasticity in compression is equal to that in tension, and amounts to 29,000,000 lb. per square inch, then:

 $A = \frac{29,000,000}{8}$ = the force in pounds per square inch needed to stretch or compress the bar 1 in.

A times .6902 = the force in pounds needed to stretch the large section 1 in.

A times .6902 times $\left(.o_{13} - \frac{3X}{4}\right)$ = the force in pounds needed to stretch the large section $\left(.o_{13} - \frac{3X}{4}\right)$ parts of an inch.

Similarly,

A times .125 times X = the force in pounds needed to compress the small section X parts of an inch.

Then,

$$A (.6902) \left(.013 - \frac{3X}{4}\right) = A(.125)(X)$$

Whence X = 0.014 in.

Then,

 $\frac{29,000,000}{8} \times (.014) = 50,750$, the stress in pounds per square inch that was set up in the small section of this casting. This figure is of course only an approximation, as the accuracy of measurement is not great, and the bars are somewhat deflected by the stresses set up in cooling. Indeed, this figure is above the elastic limit in compression, and hence an impossible one. The great severity of this stress, however, will be easily believed when it is stated that, in the case of hard and brittle steels like manganese steel in the cast state, only by cutting such a casting from its sprue can it be kept from breaking; if it is pounded with a hammer, a few blows will cause the light parts connecting the two legs to snap in two like pipe stems, and many of the castings will break apart even in cooling in the air.

These residual stresses can be relieved almost entirely by allowing the castings to cool slowly and evenly from the initial heat in a furnace—the castings either being put in at a high temperature, or got in at such temperature as can be managed, and heated up. Were the relief of stress the only, or even the chief function of annealing, therefore, it would be quite sufficient to cool the castings in pits, such for instance as are used for chilled iron car wheels, or in annealing furnaces.

We know, however, that in order to secure the greatest toughness and strength, we must heat our castings for a certain length of time to proper temperatures, and cool them from those temperatures at an appropriate rate. This annealing could be carried out without allowing the castings to grow entirely cold after solidifying; commonly, however, they are allowed to cool and are cleaned of adhering sand, and the sink heads and sprues removed, before annealing.

The difficulty and expense of properly heating them up when heavily coated with sand, and the extra expense involved in heating sink heads, make this practice essential for proper annealing.

It is very commonly assumed that the fracture of a bar of cast steel, as viewed by the eye, is sufficient examination of the "structure" to indicate whether a given piece of steel has been annealed or not, and if the annealing has been properly carried out. be stated without hesitation, however, that the fracture is a most insufficient indication of the thoroughness of the annealing, since in any piece of cast steel that has been reheated above a certain temperature, the characteristic coarse crystallization of the raw steel will be found to have been replaced by a practically uniform fine grained fracture, quite regardless of the rate of cooling from the annealing temperature. The eye, therefore, is unable to distinguish in the fracture of the steel any considerable differences in internal structure, due to variations in the rate of cooling. When we come to study the internal structure of the steel, as revealed by examining under the microscope the polished surface of a test piece etched with suitable acids, we find that in two steels whose fractures as judged by the eye are identical, there may be enormous differences in structure; and tests will soon show us that these differences in microstructure are accompanied by, and can be used as indications of, great differences in physical properties.

The science of metallography is a study in itself, has its own most extensive literature, and can be covered here only in the barest outline, as applied to cast steels. First and foremost, it must be understood that by this term is meant not the mere examination of the steel under the microscope; the science of metallography in its broadest sense includes the physical, and all other testing of steel, and the study of the effect of chemical analysis, methods of manufacture, methods of heat treatment, even methods of moulding and pouring, upon it. The microscope is but one tool, although an important one, in the hands of the metallographist.

In preparing steel for microscopic examination, one face of the specimen is first brought to a mirror surface by polishing on emery papers or cloths of increasing fineness, and on buffing wheels coated with tripoli powder or jeweler's rouge. The polished surface is next etched with acids, iodine, or other corrosive agents, that attack the microscopic constituents unequally, then washed and dried. The microscopic constituents of ordinary steel thus revealed are as follows:

Ferrite.—Pure iron, free from carbon; slightly colored by acids; the junctions of the ferrite crystals are brought out by etching agents, which eat into the junctions and reveal them as black lines. Carbonless steel or wrought iron is composed almost entirely of ferrite, but in wrought iron the slag fibers make up a considerable part of the section.

Cementite.—The chemical compound, Fe₃C, of iron and carbon. Very hard and brittle; not colored by acids, appears clear white; but colored dark by boiling sodium picrate.

Pearlite.—The eutectoid mixture of cementite and ferrite found in steel containing carbon, when cooled slowly from above the recalescence point. It is built up of thin parallel plates of ferrite and cementite alternately. Colored black by acids, when viewed under low power. Under high power, the banded appearance is seen—cementite white and ferrite black, because the ferrite is eaten away below the level of the cementite plates and does not reflect the light.

At from .80 to .90 per cent. (generally given at .89 per cent.), carbon, the entire mass of ordinary steel in the slowly cooled state consists of pearlite.

Steels between o per cent. carbon and .89 per cent. carbon are composed of ferrite and pearlite in proportion varying with the carbon. Low carbon steels are composed of ferrite with small areas of pearlite. In steel of .25 per cent. carbon the pearlite occupies about one-third of the cross-sectional area. As carbon increases, pearlite increases, the ferrite occupying less and less of the area; at about .60 per cent. carbon the ferrite appears chiefly as a net-work surrounding meshes of pearlite; as .89 per cent. carbon is approached the net-work of ferrite becomes discontinuous and the nets thin, finally disappearing. When the steel is in the ordinary annealed condition, the carbon content can be estimated quite closely by the relative proportions of ferrite and pearlite. Heat treatment, however, so affects the distribution of the constituents as to render the estimation of carbon content by this means impossible.

Above .89 per cent. carbon, free cementite appears as a net-work surrounding pearlite meshes, the net-work growing more continuous and the nets thicker, as carbon increases. Many needle-shaped cementite areas intersect the pearlite meshes at various angles.

In general, the strength of the steel rises, and the brittleness increases, in direct proportion to the carbon content.

Steels below .89 per cent. carbon are called hypo-eutectoid steels. Steels of .89 per cent. carbon are called eutectoid steels.

Steels above .89 per cent. carbon are called hyper-eutectoid steels. Sorbite.—Similar to pearlite, but cannot be resolved under high power. The compound nature of the sorbite can be discerned, but the cementite and ferrite are so minutely divided that they cannot be seen as such. Sorbite is typical of carbon steel that has been cooled in air from above the recalescence point, so that the austenite has been transformed too rapidly to produce true pearlite. Intermediate stages between sorbite and pearlite also occur in which the cementite and ferrite can be resolved under high power, but the banded structure of true lamellar pearlite is not seen. These intermediate forms are known as "emulsified" pearlite, and merge into lamellar pearlite at one end of the scale, and into sorbite at the other.

Austenite.—The solid solution of Fe₃C, in iron; above the critical range, the entire mass of a steel exists as austenite. Cooled with great rapidity from above the critical range, as for instance, by quenching a thin piece of steel in iced brine (particularly if much carbon, nickel or manganese is present), steel consists wholly or partly of austenite.

Colored yellowish by acids; appears as uniform grains of a single substance, resembling ferrite; the crystal junctions outlined as black lines.

Martensite.—Austenite partly transformed; a transition product between austenite and sorbite. Very hard and brittle. Etched with iodine exhibits markings in acicular or arrow-head arrangement, sometimes heavily outlined in black. Characteristic of carbon steel that possesses marked hardening power, and that has been quenched from above the recalescence point.

Critical Points.—If a sample of steel be heated or cooled in a furnace whose temperature is rising or falling at a uniform rate, with a thermo couple inserted in the piece of steel, it will be found that at certain well-defined temperatures, marked changes occur in the rate at which the temperature of the steel is rising or falling. These "critical points" as they are called, in the heating or cooling curves of the steel are due to the sudden absorption or evolution of heat caused by changes of state of one or more constituents of the steel. That which occurs at the lowest temperature is known as the "Rescalescence Point," because at that temperature, steel containing enough carbon to form a fair proportion of pearlite, evolves sufficient heat in cooling to cause its temperature to remain stationary for some moments, or even to rise slightly; in heating, the change

from pearlite to austenite absorbs so much heat that for an appreciable interval the temperature of the steel does not rise. This phenomenon is known as recalescence.

Two of the critical points, A_1 and A_3 , occur at somewhat higher temperatures in heating than in cooling. This phenomenon, which is really the raising or lowering of the critical point from the temperature at which it should occur, is due to lag, or molecular inertia—the fact that the transformations require time to complete themselves. Hence the raising or lowering of the temperature is the greater, the more rapid the rate of heating or cooling. Could that rate be made infinitely slow, the critical points in heating and in cooling would be at the same temperature. As it is, the temperature interval is considerable, so that we have to distinguish between the two points by the symbol Ac for heating and Ar for cooling. Normally Ar_1 , for instance, occurs at 680 or 690°C., and Ac_1 at 720 to 740°.

It is this dragging down of the transformation points by rapid cooling, assisted by the influence of carbon, manganese, nickel and some other alloys, that gives steel its hardening power. Very rapid cooling so far depresses the temperatures at which the transformations take place that they are wholly or partially suppressed, giving us austenite in the former case, and martensite or sorbite, in the latter. This discussion omits troostite and some other transition constituents that are not of great importance in the heat treatment of steel castings; and omits also some other means by which the transition constituents can be obtained, as for instance, quenching steel heated above the recalescence point, but below the other critical points.

The Carbon-iron Diagram.—The carbon-iron diagram (Fig. 27) shows the temperatures at which steels of varying carbon content solidify, and the temperatures at which their critical points occur, plotting temperatures against carbon per cent. With all that portion of the diagram lying to the right of the point, carbon equals 1.75 per cent. we shall not concern ourselves in this discussion, as the very high carbon steels and cast irons are of little interest to the steel foundryman.

Referring to the diagram, the line PSK is the recalescence point A_1 at which pearlite is formed on cooling, and changes to austenite on heating (called Ac_1 , heating, Ar_1 , cooling).

MO is the point A_2 (Ac_2 heating, Ar_2 cooling), at which alpha ferrite becomes beta ferrite on heating, and vice versa.

GOS is the point A_3 (Ac_3 heating, Ar_3 cooling), at which beta ferrite is liberated from austenite on cooling, or changes to gamma ferrite and dissolves in austenite on heating.

It will be seen that above about .35 per cent. carbon, A_3 falls until A_2 and A_3 unite, becoming a single point A_{2-3} .

 A_{2-3} continues to fall as .89 per cent. carbon is approached, until at .89 per cent. carbon, the transformations all take place at the

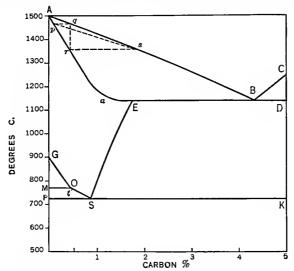


Fig. 27.—The carbon-iron diagram. From Proceedings of Sixth Congress, International Association for Testing Materials.

recalescence temperature A_1 , giving us a single point A_{1-2-3} .

SE represents the temperature at which free cementite in steel above .89 per cent. carbon is dissolved in austenite on heating, and is set free on cooling.

In crossing the line MOSK in heating, iron loses the bulk of its magnetism; and, of course, recovers it on recrossing the line as it cools.

At the line AB, solidification of liquid steel takes place. The line Aa represents the temperature and composition of the solidfying metal, the line AB, that of the molten mother metal. The first freezing particles are of lower carbon content than the mass of the steel. Thus, a steel of .50 per cent. carbon begins to freeze at the point (q), and the composition of the first freezing layers is (p). As freezing proceeds, the molten mother-metal becomes progressively

richer in carbon, its temperature and composition sliding along the line AB, and the successive particles of freezing metal also become richer in carbon, their temperature and composition sliding along the line ps. Meanwhile diffusion of carbon begins in the frozen portions of the mass, tending to equalize the composition of the successive layers. If this diffusion is complete, the temperature and composition of the frozen portion slide along the line pr.

When the temperature of the steel has fallen to Aa, the final particles freeze. Thus a .50 per cent. carbon steel is completely frozen at a temperature represented by (r); the temperature and composition of the last freezing particles will be (s); that of the last particles of the molten mother-metal to freeze, also (s), and if diffusion has been complete in the frozen portions, the composition and temperature of these portions will be (r).

At any temperature between (q) and (r), on the line (qr), the amount of solid is proportional to the horizontal distance between this point and the line AB, the amount of liquid to the horizontal distance from this point to the line Aa. Thus when we reach the temperature (r), no liquid remains, and the composition and temperature of the last freezing particles have reached (s). If diffusion were complete, these last freezing particles would then assume the composition (r), of the frozen portion.

Actually, diffusion is not complete, so that the first frozen particles contain less carbon than those last frozen. The austenite of the just frozen steel is thus heterogeneous, the first freezing particles forming a net-work of low carbon austenite, the axes of the nets being lowest in carbon.

During the cooling of the steel through the region AaESOG, further diffusion occurs; and if time enough were allowed for diffusion to complete itself, the initial heterogeneity of the steel would be effaced. With ordinary rates of cooling, however, the net-work of low carbon austenite (primary austenite), remains to form primary ferrite when the line GOS is crossed.

On reaching temperature (t) on the line GOS, free ferrite begins to be liberated from the (solid), austenite, the composition of the remaining mother austenite as the temperature falls sliding along the line OS until it reaches the composition S, when the remaining austenite transforms to pearlite, with the accompanying evolution of heat. In the liberation of free ferrite as the steel cools from GOS to PS, those parts of the steel lowest in carbon liberate their ferrite first, because their transformation point is highest; and thus the pri-

mary austenite net-work is perpetuated as a net-work of (primary) ferrite, which is made visible by etching. The (secondary) ferrite set free within the meshes of austenite forms finer net-works and veins.

Referring to Fig. 29, a cast steel showing typical net-work ferrite, it will be noticed that the heavy white lines are not arranged in a

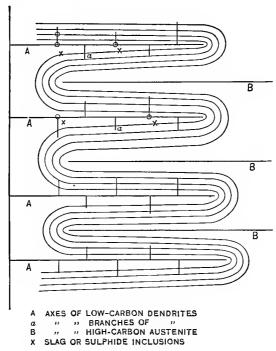


Fig. 28.—Dendritic freezing upon nuclei of slag or sulphide of manganese.

true net-work, but rather are roughly parallel. The inclusions of slag or sulphide of manganese are located at the axes of these ferrite lines. This suggests that in the original solidification of this steel, the austenite that froze first assumed the pine-tree or dendritic shape generally characteristic of the first-freezing crystals. In such a method of crystallization, the austenite shoots out into the molten metal in long sheets, with parallel side branches. If there is anything in the theory of nucleus action, it is reasonable to assume that these dendrites may be formed upon slag or sulphite droplets as nuclei. According to this reasoning, the just frozen steel would be heter-

ogeneous in the manner shown diagrammatically in Fig. 28. The lines Aa represent the low carbon axes of the first-freezing austenite dendrites, the lines B, the corresponding axial lines of the highest carbon austenite rejected, during the process of freezing, into the spaces between the dendrites. It must be understood that the carbon content increases progressively from A to B; and that diffusion will partly eliminate the smaller side branches aa, and also, of course, lessen the difference in carbon content between A and B.

When the transformation range A_3 is reached, the low carbon axes A will liberate their ferrite first, because the liberation of ferrite takes place at a higher temperature, and therefore sooner, in the lower carbon portions of the steel. If this ferrite tends to precipitate upon the slag or sulphide inclusions, as it is reasonable to assume it will, the ferrite will largely remain in the position of the axes of the original dendrites of primary austenite; and perhaps will concentrate strongly around the inclusions. The ferrite rejected from the space between A and B will to some extent follow the first precipitating ferrite, and no doubt will concentrate upon the inclusions.

The greater the time consumed in crossing the transformation range A_3 to A_1 , the more thoroughly will the ferrite be concentrated from this intervening space. Thus, in the comparatively small casting shown in Figs. 29 and 31, there is a great deal of ferrite precipitated in this intervening space; while in much heavier castings, such as shown in Fig. 34, especially in the interior where the cooling is slowest, the concentration goes further, producing large lakes of ferrite about the slag inclusions, and leaving the intervening space much freer from ferrite. The coarseness of the original heterogeneity of such a large steel casting reinforced by this transformational segregation, makes these ferrite masses very large.

The researches of Professor Howe¹ show that in the formation of ferrite at A_3 , the net-works are first formed, then a great deal of ferrite is liberated within the meshes of austenite, much of it being located in the octahedral cleavages of the austenite crystals; then, if the cooling is slow enough, this ferrite migrates to the networks, thickening them, and decreasing the amount of ferrite within the grains. In certain cases, the ferrite in the octahedral cleavages is so abundant as to mask the net-works, and occasionally the latter appear to be lacking. The evidence at hand appears

¹A. S. T. M., 1911, p. 263. International Journal of Metallography, vol. iii, p. 4, and elsewhere.

to show that in rolled or forged steels, a long, high heating (above A_3) so increases the tendency of this cleavage ferrite to persist that it is not absorbed into the net-works. Balaiew¹ obtained this structure, associated with a net-work, in a steel cooled very slowly from the solidification point. It is characteristic of meteorites; and has been found in masses of steel that are known to have been heated for a long time to an elevated temperature.

It is possible that the absence of slag or sulphide nuclei, upon which primary austenite can precipitate, may in some cases so retard the precipitation of that austenite in freezing that the initial heterogeneity of the steel is not as great as in the majority of cases. Should this tendency to restrain initial heterogeneity be reinforced by a cooling slow enough to allow diffusion to make the austenite quite homogeneous, the ferrite might be so largely precipitated at A_3 in the cleavages of the austenite grains, by a suitable rate of cooling through the transformation range, as to give rise to this structure to a very great extent. At any rate, the presence in cast steel of this Widmanstättian structure, as it is called, is often accompanied by a marked lack of slag or sulphide inclusions. Figure 30 shows a typical example of this structure.

It may be that the amount of slag or sulphide inclusions may have a determining influence in fixing the size of the ferrite networks in steels that show a net-work structure. Thus the presence of but few inclusions, especially if they are large, may concentrate the primary austenite at a few points, and so give rise to a coarse net-work; and that of a great many small inclusions, to a fine network. If the location of the net-work ferrite is governed in part by the position of the primary austenite net-works, and if the ferrite set free at A_3 has a tendency to precipitate upon the inclusions, they may well have this effect upon the size of the structure.

The size of the casting, however, is the chief factor in determining the net-work size. This is partly, no doubt, because the austenite net-works formed in the slow freezing of the large casting are coarse; and partly because the migration of the cleavage ferrite into the net-works, and that of the smaller net-works into the larger ones, during the cooling from A_3 to A_1 , has time to proceed very far, on account of the slow cooling of such large castings. The interior of heavy castings, moreover, exhibits to a marked extent the "balling up" of the net-work ferrite, and its concentration into great lakes (often surrounding slag or sulphide inclusions), which, as Professor

¹ Revue de Metallurgie, 1910, p. 510; 1912, p. 321.

Howe has shown in the articles referred to above, takes place to a marked extent during slow cooling.

The prime object of annealing or heat treatment is to break up by heating the coarse crystallization of ferrite and pearlite (ingotism, or ingot structure), and replace it by a uniformly fine crystallization of the constituents, with resulting strong inter-crystalline bonding; and the abolition of the long planes of inter-crystalline weakness due to the large sheets of ferrite in the slowly cooled steel. Heating the steel above AC_3 , that is, above the line GOS, causes the pearlite to transform to austenite (in crossing AC_1), and the ferrite to dissolve in the austenite (in crossing AC_3). The absorption of the ferrite, however (especially the primary ferrite), requires time, since it necessarily consists in the diffusion of carbon from the high carbon into the low carbon portions of the steel (a microscopic cementation, as it were, of the ferrite by the pearlite portions), and cementation we know is a gradual process.

The coarseness of the microstructure of the steel, as we have seen. depends chiefly upon the size of the casting, which determines the rate at which it has frozen and cooled. It would seem, therefore, that the time necessary for a fairly complete diffusion of the carbon at a given temperature, would depend upon the size of the castings, as one would suppose that the larger the ferrite areas the longer it would take to cement them. Work done by the author, however. has shown that it takes no longer to cement the large ferrite areas in the center of a casting 6 in. thick, than the small ones in a piece $\frac{1}{2}$ in, thick. In this research, pieces about $\frac{1}{2}$ in, cube were cut from $\frac{1}{4}$ in, square bars, and from the outside and center of 6-in. square bars, of .25 C, .40 C, and .55 C steel respectively. These small pieces were annealed together at 800°C. for ½ hour, 1 hour and 2 hours, and at 900°C. and 950°C. for the same lengths of time. After annealing they were cooled in lime, which in such small pieces resulted in a comparatively rapid cooling, and a correspondingly fine-grained miscrostructure. Examination of the 90 micrographs made developed the surprising facts that in many cases the $\frac{1}{2}$ in. square bars showed more free ferrite (due to under-annealing) after 2 hours at 800°C. than did the pieces cut from the outside of the 6-in. square bars; and that the free ferrite of the $\frac{1}{2}$ in. square bars is absorbed at about the same rate as the much larger masses of ferrite at the center of the 6-in. square bars. The 12 micrographs. Figs. 31 to 42, show the $\frac{1}{2}$ -in. square bars, and the center of the 6-in. square bars of the .25 C, and of the .55 C steel, in the cast condition,

as annealed at 800°C. for 2 hours, and after 2 hours annealing at 900°C. These micrographs show quite clearly that the 6-in. square bars anneal no more slowly than the ½-in. square bars. In a paper1 by Messrs. Knox Taylor, Arvid E. Nissen and the author, a number of other micrographs from this same research have been published, which prove this even more conclusively than the twelve illustrations selected for the present volume. It is clear from this evidence also that heating to 900°C. for 2 hours is suitable for annealing steel of .55 C, as well as steel of .25 C. Theoretically, the higher carbon steels should be annealed at lower temperatures, to avoid undue coarsening of the grain; but the micrographs show that ooo°C. is as suitable for the high carbon as for the low, as there is no coarsening of the grain at that temperature. Thus although the carboniron diagram shows that .60 per cent. carbon steel can be annealed at a lower temperature than .20 per cent. carbon steel, yet the rate of diffusion is so much lower at the lower temperatures, that for practical purposes it will in many cases be found convenient, for steel normally containing free ferrite, to use a uniform annealing temperature of from 850 to 900°C.

The castings should be heated to this temperature for at least 2 hours and preferably 4 hours, after they have become hot through—very large castings may be heated longer if it is found necessary. The total annealing time of the large castings will in any case be considerably longer than that of the small ones, as it requires so much longer to bring them up to heat throughout their mass. Roughly, an exposure to the annealing temperature of $\frac{1}{2}$ hour per inch of section of the casting is required to heat the steel to the center.

The use of much higher annealing temperatures and shorter time has occasionally been suggested, but the resulting expense in fuel and furnace wear is prohibitive, and the castings scale and warp badly. Moreover, the microstructure of the steel is greatly coarsened at these higher temperatures, so that a second annealing at between 800 and 900°C. would have to be resorted to if really good steel were desired.

Now, upon cooling from the annealing temperature, these initially coarse-grained cast steels have a strong tendency to reprecipitate their ferrite in heavy masses, apparently located to a great extent along the lines of the old primary ferrite net-work. Steels which possessed only the Widmanstättian arrangement of the ferrite

1" The Heat Treatment of Cast Steel," Proc. Am. Inst. Min. Metall. Eng., vol. lxii (1920), p. 353.

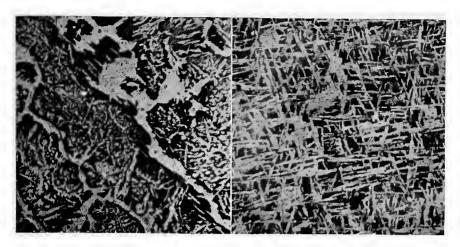


Fig. 29.-0.43C-as cast.

Fig. 30.-0.37C-as cast.

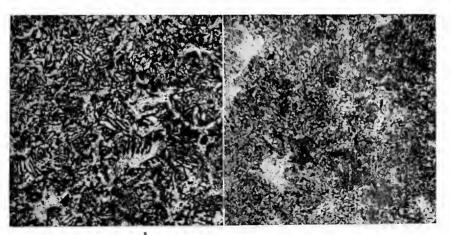


Fig. 31.—0.29C—1/2 in. square bar—as cast.

Fig. 32.—Same as Fig. 31—heated to 800°C, for 2 hours and cooled in lime.

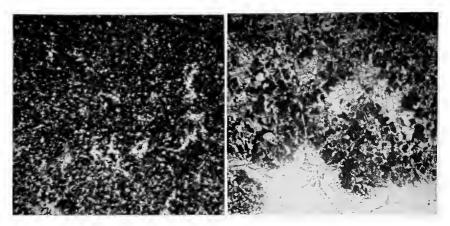


Fig. 33.—Same as Fig. 31, heated to 900°C. Fig. 34.—0.25C.—center of 6 in. square for 2 hours and cooled in lime. bar—as cast.

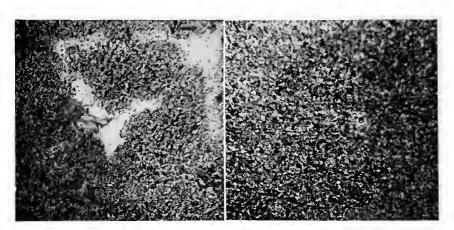


FIG. 35.—Same as Fig. 34, heated to 800°C. FIG. 36.—Same as Fig. 34, heated to 900°C. for 2 hours and cooled in lime.

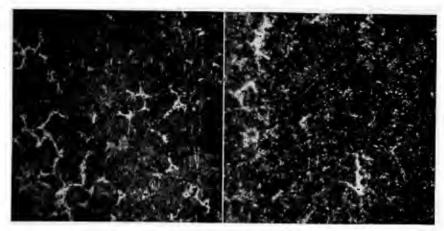


Fig. 37.—0.55C—½ in. square bar—as cast.

Fig. 38.—Same as Fig. 37, heated to 800°C. for 2 hours and cooled in lime.

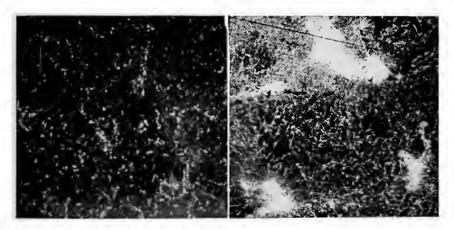


Fig. 39.—Same as Fig. 37, heated to 900°C. for 2 hours and cooled in lime.

Fig. 40.—0.56 C—center of 6 in. square bar —as cast.

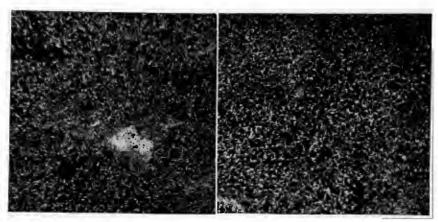


Fig. 41.—Same as Fig. 40, heated to 800°C. for 2 hours and cooled in lime.

Fig. 42.—Same as Fig. 40, heated to 900°C. for 2 hours and cooled in lime.

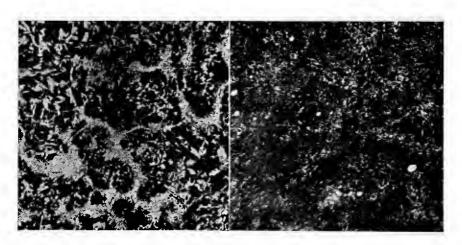


Fig. 43.—o.43C, annealed at 900°C., quenched at 800°C.

Fig. 44.—0.39C, annealed at 900°C., quenched at 750°C.

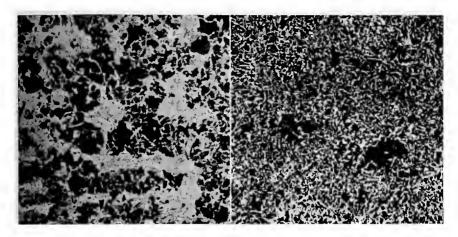


Fig. 45.—0.41C, annealed at 900°C. and cooled slowly.

Fig. 46.—0.43 C annealed at 900°C., cooled in 10 minutes.

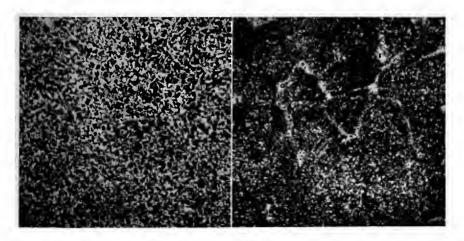


Fig. 47.—0.41C, annealed at 900°C., cooled in 40 minutes.

Fig. 48.—0.39C, annealed at 900°C., quenched at 800°C., reheated to 7 00°C. and quenched.

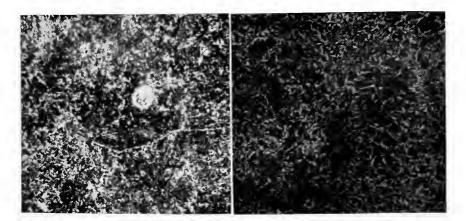


FIG. 49.—0.44C, annealed at 900°C., quenched at 800°C., reheated to 650°C. and quenched.

Fig. 50.—0.48C, annealed at 900°C and air cooled. Reheated to 700°C, cooled slowly.

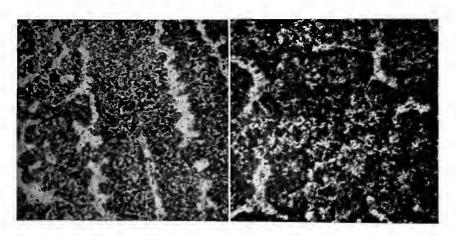


FIG. 51.—0.51 C, annealed at 900°C. and air cooled. Reheated to 760°C., cooled slowly.

Fig. 52.—0.59C, 0.023S, 0.017P, annealing similar to Fig. 51.

appear on slow cooling from the annealing temperature to show a ferrite-pearlite structure of fair uniformity, though not fine grained; but steel that was of the net-work class shows ferrite in very coarse masses. Reference to the accompanying microphotographs illustrates this very clearly.

Figures 29, 30, 31, 34, 37 and 40 show the characteristic micro-

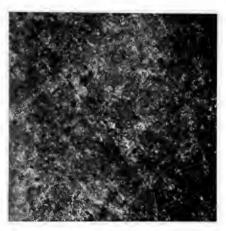


Fig. 53.—0.21C, 1.17Mn, quenched and reheated.



Fig. 54.—About & full size.

Fig. 55.—About \(\frac{1}{3} \) full size.

structures of medium carbon steel in the cast condition, Fig. 29 being of a typical net-work structure steel, Fig. 30 of a steel in which the Widmanstättian structure predominates.

Figure 43 shows a similar steel annealed for 2 hours at 900°C. cooled slowly to 800°C. and quenched in water; and Fig. 44 a steel annealed for 2 hours at 900°C., cooled slowly to 750°C. and quenched in water. Figure 43 shows a very uniform sorbitic structure (the cooling being too slow, owing to the size of the test bar, and the

| No. | С | Si | Mn | S | P | | | |
|------------|------|------|--------------|------|-------|--|--|--|
| 29 | . 43 | . 38 | . 47 | | | | | |
| 30 | .37 | .32 | .51 | | | | | |
| 31, 32, 33 | . 24 | .33 | . 49 | | | | | |
| 34, 35, 36 | . 25 | . 38 | .43 | | | | | |
| 37, 38, 39 | . 55 | . 56 | . 76 | | | | | |
| 40, 41, 42 | . 56 | .42 | .75 | | | | | |
| 43 | .42 | .51 | . 60 | | | | | |
| 44 | . 39 | . 48 | .47 | | | | | |
| 45 | .41 | .45 | .51 | | | | | |
| 46 | 43 | .49 | . 6 o | | | | | |
| 47 | .41 | . 52 | . 68 | | | | | |
| 48 | - 39 | .41 | - 55 | | | | | |
| 49 | - 44 | .35 | . 66 | | | | | |
| 50 | . 48 | . 54 | . 80 | | | | | |
| 51 | . 51 | .42 | . 60 | .057 | . 049 | | | |
| 52 | . 59 | . 14 | .72 | .023 | .017 | | | |
| 53 | . 21 | | 1.17 | | | | | |
| 54, 55 | . 20 | .45 | I.02 | .057 | . 049 | | | |

Analyses of Steel Shown in Figs. 29-54

carbon too low, to produce martensite), while in Fig. 44, as the steel has been allowed to cool slowly to a temperature below Ar_3 , the ferrite has reprecipitated, and shows a marked tendency to the net-work arrangement. This tendency of the ferrite to coarse crystallization is also very strongly marked in Fig. 45, a similar steel annealed at 900°C. for 3 hours and cooled very slowly; 6 hours were spent in cooling this bar from 900°C. to 600°C. There is more ferrite in Fig. 45 than in Fig. 44, because in the latter the separation of ferrite that normally takes place in cooling from 750°C. to 680° has been partially suppressed by quenching.

The black areas in Fig. 44 are sorbite, because the rapidity of the final cooling has prevented the formation of true (lamellar) pearlite; in Fig. 45, the black areas are pearlite. Higher magnification should show this clearly.

Figure 46 shows the structure of a similar steel annealed at 900°C. for 3 hours, and cooled in the air in about 10 minutes. Here, evidently, we have quite a different structure from any that we have yet seen. The black sorbite remains, and in two places it shows a strong tendency to dominate and give a structure like that of Fig. 43, but the cooling, although not sufficiently rapid to repress altogether the liberation and agglomeration of ferrite, yet has not been slow enough to allow the ferrite to agglomerate to such an extent as to leave large ferrite areas. Hence, we have the comparatively

fine ferrite net-work, resembling somewhat a rolled steel after annealing.

Figure 47 shows a similar steel heated to 900°C. for 1 hour, and cooled to black in about 40 minutes; the cooling through the critical range has been too rapid again to permit the formation of large ferrite patches, though, as will be seen by comparing Fig. 46 and Fig. 47, the coagulation of the ferrite has partially taken place, since the ferrite in Fig. 47 is noticeably coarser than that in Fig. 46. The white areas in Fig. 47 are ferrite, the black, sorbite or pearlite.

The formation of lakes of ferrite which has been prevented by quenching as shown in Fig. 43, or by air cooling as shown in Fig. 46, will take place again if the steel is reheated to Ar_1 , or nearly to that point. Curiously enough, the reappearance of ferrite in a coarse net-work takes place at a lower temperature in the reheating of a quenched steel than in the reheating of an air-cooled steel.

Thus, Fig. 48 shows a steel of this class annealed at 900°C. for $2\frac{1}{2}$ hours, cooled slowly to 800°C. and quenched (resulting of course in structure like that of Fig. 43), than reheated to 700°C. for a few minutes and again quenched. The separation of very finely divided ferrite from the sorbite background is clearly shown, and also the agglomeration of ferrite into lakes in a partially defined coarse net-work. Figure 49 shows a similar steel annealed at 900°C. for $2\frac{1}{2}$ hours and quenched in water, then reheated to 650°C. and again quenched. No agglomeration of ferrite into lakes can be noticed.

Figure 50 shows a similar steel annealed at 900°C. for 4 hours and cooled in air, then reheated to 700°C. for 4 hours and cooled slowly; and Fig. 51, a steel heated to 900°C. for 4 hours and cooled in air, then reheated to 760°C. and cooled slowly. Comparing Fig. 50 with Fig. 46, we see that the structure of the air-cooled steel has been little if at all altered by the reheating to 700°C., a temperature which brought out coarse ferrite in the quenched steel. In Fig. 51, however, there is an agglomeration of ferrite over the whole background, and also a well-defined agglomeration of ferrite into a large net-work.

These microphotographs have been selected from some hundreds which were made in the course of an extended research into the behavior of cast steel in heat treatment, and are typical examples. No hypo-eutectoid carbon cast steel tested in this research, and none yet brought to the author's attention, has failed to behave in the characteristic manner shown by these photographs. To be sure, the tendency to the formation of coarse ferrite showing traces of the net-

work arrangement is not equally marked in all cast steels, yet none are wholly free from it. The predominance of a Widmanstättian arrangement of primary ferrite in the raw steel appears to decrease or mask the formation of this coarse net-work after annealing; and the presence of many slag or sulphide inclusions (usually associated with a strongly marked net-work structure in the raw steel), to increase it.

All electric furnace steels, however low in sulphur and phosphorus, that have come to the author's attention, show this same tendency to the retention of coarse ferrite patches after annealing; some more than others. Figure 52, an electric furnace steel containing but .023 per cent. sulphur and .017 per cent. phosphorus, shows that even a steel very low in phosphorus and sulphur exhibits this tendency very strongly. The rather numerous inclusions in this specimen must be largely slag, which suggests that slag resists the advance of carbon into the ferrite as strongly as does sulphide of manganese, which would naturally be the case if the inclusions act as mechanical barriers to the progress of the diffusing particles; that slag as well as sulphide of manganese particles tend to hold the ferrite because it adheres to them; or that, as already suggested, the action of the inclusions is indirect, that is, that they act as nuclei upon which net-work austenite is deposited in freezing, and so favor the formation of coarse ferrite net-works, which are eliminated slowly by diffusion in annealing. This steel was annealed at 900°C. for 4 hours and cooled in air, then reheated to 760°C. for 2 hours and slowly cooled. The well-marked coarse net-work is clearly shown, though it is less noticeable than in the previously exhibited examples, because, owing to the higher carbon (.50 per cent.), there is much less available free ferrite in this steel to form the coarse patches.

Some work done recently on electric furnace steels extremely low in phosphorus and sulphur, and reported in the paper before the American Institute of Mining and Metallurgical Engineers referred to above, indicates that in these steels the tendency of the ferrite to reappear in coarse patches if the second heating of a double anneal is carried to too high a temperature is considerably less marked than in the Bessemer steels on which most of these experiments were carried out.

Even in these extremely pure electric steels, however, an annealing at 900°C., followed by slow cooling produces a coarse ferrite-pearlite miscrostructure similar in every way to that of the Bessemer steels. It is plain, therefore, that to secure in any carbon cast steel the

benefits due to a fine microstructure, comparatively rapid cooling followed by reheating to a proper temperature must be resorted to.

The great majority of steel castings requiring annealing are heated to about 900°C. and after soaking are slowly cooled, resulting in a structure similar to that of Fig. 45; coarser in larger castings, finer in smaller ones. In some cases, by opening the furnaces more or less and hastening the cooling, structures like Fig. 46 and Fig. 47 are obtained. The annealed steel will have a higher tensile strength, elastic limit, extension, contraction of area, and bend than the steel in the raw state; and by accelerated cooling the strength and elastic limit can be raised without sacrifice of toughness, or with actual gain of toughness; accelerated cooling, in particular, increases the resistance to sudden shock. The annealed castings will be softer than the raw steel—and, of course, quite free from shrinkage stresses, unless the accelerated cooling is continued until the castings are cold.

If accelerated cooling is used, the furnace should be closed up after the castings have reached black heat, and the cooling finished slowly enough to allow of the relief of cooling stresses. If the accelerated cooling has been very rapid, or if an air or water cooling has been used, the stresses set up in the castings will be nearly as great as in the raw state, and they must be heated a second time to relieve the stresses.

For convenience, the rates of cooling from the annealing temperature may be classified as follows:

- 1. Slow—over 50 minutes from 900°C. to black.
- 2. Accelerated { over 10 minutes from 900°C. to black. less than 30 minutes.
- 3. Rapid { less than 10 minutes from 900°C. to black, as for instance cooling in air or air blast.
- 4. Very rapid (quench in oil or water).

It should be obvious from what has been said that to give steel castings an annealing followed by slow cooling, as a preliminary to a double heat treatment, is quite unnecessary and a waste of time and money; for during the slow cooling, the ferrite and pearlite will separate into a structure nearly as coarse as that of the untreated cast steel. The usual object of a preliminary annealing, a refinement of the grain which will allow the first heating of the heat treatment proper to be made quite brief, is not attained in cast steel, as it is in forged material. The second heating, then, in the case of the casting, must be to about the same temperature and for as long

a time, as the preliminary annealing, in order to obliterate this coarse structure, and thus the two heatings perform the same function, and the first one is quite wasted.

GENERAL RULES

General rules for the heat treatment of hypo-eutectoid carbonsteel castings may be given as follows:

Heat to 850 or 900°C. for from 2 to 4 hours after the castings are hot through.

The more rapid the cooling from the annealing temperature, the finer will be the microstructure, and the greater will be the hardness of the steel, and the stresses resulting from unequal cooling of sections of unequal size. Rapid coolings always, and accelerated coolings sometimes, must be followed by a second heating to a lower temperature to reduce hardness and relieve stresses. Accelerated coolings should be followed at least by a period of slow cooling, as, for instance, by closing the furnace again after the castings have cooled to black heat.

The temperature used for reheating after rapid cooling should not exceed 680°C., if the castings have been quenched in oil or water, and should not exceed 700°C., if the castings have been cooled in air or air blast. The castings should be held at these temperatures a sufficient length of time to produce the softening effects desired. Quenched castings, which are given the maximum toughness by reheating to from 640 to 680°C., should be held at the reheating temperature from 2 to 8 hours. Air-cooled castings, which for maximum toughness should be reheated to from 660 to 700°C., should be held at the reheating temperature from 2 to 6 hours. The lower the reheating temperature and the shorter the reheating time, the higher will be the strength and elastic limit of the steel, and the less the toughness, as measured by extension, contraction of area, bend and shock test. Reheating to temperatures that liberate coarse ferrite causes great loss of strength and toughness.

In general, the more rapid the cooling from the annealing temperature, if proper reheating temperature and time have been used, the greater will be both strength and toughness—these properties bearing a direct relation to microscopic fineness of structure. More especially, the shock toughness of soft and medium soft steel is greatly increased by rapid cooling and proper reheating.

The maximum softness, and the maximum magnetic permeability,

however, are attained by very slow cooling from the annealing temperature, which results in the liberation of the maximum amount of free ferrite. Steel so annealed is but little softer as measured by the Brinell or scleroscope test, than steel cooled in air, oil or water and reheated to the proper temperature for maximum toughness; but the tougher heat treated steel, simply because it is so tough, cannot be machined as rapidly as the steel annealed by slow cooling. The chief reason for this is that the chips from the heat-treated steel do not break up into fragments but come off in long "curls" which often have to be broken from the work at intervals.

In selecting a method of heat treatment for a given casting, careful attention should be given to the size and intricacy of the piece, and especially to the possibility of cracking it if quenching is attempted. Frequently also, it will be impracticable to remove a casting from the furnace while hot, in order to quench it or cool it in the air, on account of the danger of bending it seriously. In such cases, accelerated cooling by opening all doors, etc., of furnaces, followed by proper reheating, will give good results.

When the castings contain such heavy sections that they will not cool to a black heat in less than 30 minutes, either in the air or in the open furnace, slow cooling should usually be resorted to; air cooling will not be of value, since it will not produce a fine microstructure, and the only benefit to be derived from accelerated cooling to black heat followed by slower cooling to relieve stresses will be the time saved in the process. These castings that are too heavy to cool to black heat in less than 30 minutes, too, are in general too heavy to quench in oil or water without great risk of causing cracks. Double heat treatments, then, are not for the maker of really heavy castings. As in most cases these castings can be given proper strength by increasing the sections, minimum weight not being an essential, plain annealing followed by slow cooling is plenty good enough.

One objection to double treatment that is quite firmly fixed in the minds of a great many foundrymen is that they suppose it requires a longer time and costs more than plain annealing. With car bottom furnaces, or some similar arrangement, however, air cooling can be carried out at the expense only of pulling out the car, and even the handling cost of quenching is not very great. The two heatings, of course, require more fuel than a single heating. Against these extra items of cost, however, should be set the saving of time secured by double heat treatment. In practically all cases, the second

heating can be followed by cooling in the air, as the stresses set up in cooling from these low temperatures are comparatively slight. The furnace, then, can be emptied almost immediately after the second heating is completed, without waiting for the furnace and castings to cool down. In actual practice, a great deal more tonnage can be put through an annealing furnace in a given time using double treatment than is possible with "old fashioned" annealing.

The maximum allowable reheating temperatures are those which effect the softening and relief of stresses in the least possible time, without resulting in the liberation of free ferrite in heavy masses. For the greatest toughness, the maximum time quoted must be used. That no harmful stresses are produced by air, or even water, cooling from the reheating temperature, has been proved by careful tests on stress test bars, and on full-sized castings of various shapes. That no hardening is thus produced has been carefully proved by experiment, and it is to be expected, since hardening results only on cooling steel after it has been heated above Ac_1 , and the reheating temperatures given do not exceed Ac_1 .

The author's experience of 17 years in the heat treatment of cast steel has shown conclusively the falsity of the widely held opinion that rapid cooling, and especially quenching, of steel castings, is almost sure to result in dangerous stresses and many cracked and broken pieces. Of course, very heavy sections, especially of hard steels, cannot be quenched without rupture. To quench successfully a solid spherical or cylindrical casting 8 or 10 in. in diameter, is well-nigh impossible. But if the steel is not too high in carbon, castings with quite widely varying sections, in all weights from a few pounds up to at least a ton, can be quenched in oil or even in cold water without the loss of a single piece, especially if they are not allowed to grow dead cold in the water. In the reheating, of course, the furnace men must take pains to heat these quenched castings up slowly until they reach a temperature of 300 or 400°C.. but the work presents no difficulties not easily mastered by men familiar with the handling of high-grade steels. The author has known of cases where heavy locomotive frames of cast steel have been successfully quenched by leaving them in the water only until they had cooled to a black heat, when they were at once transferred to a furnace heated to approximately the temperature of the castings when taken from the water, and given the second heating. allowed to "go dead cold" in the water, they cracked in the heavier sections. This practice has been successfully applied to castings of simple shape in which the maximum sections are about 6 in. by 8 in. The handling of such a job, of course, is expensive, but where really strong, tough steel is required, the cost is much more than repaid by the results secured.

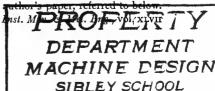
The Value of Heat Treatment of Hypo-eutectoid Cast Steel.—A test of some kind to reveal brittleness under shock is of great value in judging the effect of heat treatment. There are a number of impact testing machines now on the market, most of which use a notched test piece of small cross-section, which is broken by a heavy blow from a falling weight or pendulum provided with a knife edge to come in contact with the test piece. The force consumed in breaking the test piece is the measure of the brittleness of the steel, and the deflection of the test bar before rupture is a further indication of value. The force recorded is the sum of that necessary to effect the elastic deformation, and that necessary to effect the plastic deformation and rupture of the test piece. Because the plastic deformation of hard steels is slight (that is, they break without bending very far), while that of soft steels is very great, the total amount of force needed to break a soft steel and that required to break a hard steel in this test, are quite disproportionate. The test, therefore, should not be used to compare steels of varying carbon content. But to ascertain the toughness of a particular steel, as affected by heat treatment, it is of great value.

It has been suggested that by measuring the angle of bend in this test, a basis may be found on which to subtract the work done in overcoming plastic deformation from that expended in overcoming elastic deformation and affecting rupture, and thus that a means may be found of using the test to compare different steels.

Fremont's machine tests a bar about $\frac{1}{4}$ in. by $\frac{3}{8}$ in. by 1 in., notched with a hack saw, and broken by the blow of a weight of 10 kg. falling 4 meters. Charpy's testing machine is of the pendulum type. Several other designs are on the market.

In a paper² presented at the 1913 fall meeting of the American Institute of Mining and Metallurgical Engineers, the author has pointed out what, at least in his opinion, is the weak point of this method of testing as applied to steel castings; that is, the considerable variation in the value of the impact test from different parts of the same coupon of heat-treated cast steel. Two conclusions can

¹Prof. Henry M. Howe, Discussion of the suther's paper, referred to be ²" Shock Tests of Cast Steels," Proc. Am. Inst. Market 1913, p. 482.



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be drawn from these variations: First, that the test is not reliable as an indication of the average toughness of heat-treated cast steel in any but very light sections; second, that the maximum toughness obtainable by quenching and reheating mild cast steel is confined to the outside $\frac{1}{2}$ in. or so of thicker sections. It can be stated, however, that larger scale tests, such as are mentioned a little further on, confirm the evidence of the shock tests and show that cast steel is at its toughest and strongest when quenched and annealed, and at its worst when in the cast state; the shock toughness of the steel annealed by heating and slow cooling being considerably less than that of the heat-treated metal. Thus, though the shock test may not be quite accurate in telling us exactly how tough our steel is throughout its cross-section, yet the error is one of degree only, not of kind.

Experiments carried on for a period of years have shown conclusively that the presence of coarse ferrite, especially if it exists in a net-work structure, or in a partially developed net-work structure, whether formed on slow cooling from the annealing temperature, or on reheating after rapid cooling, to temperatures that allow the liberation of such ferrite, is always accompanied by more or less brittleness of the steel; and this brittleness is particularly marked when the steel is tested under sudden shock. This statement applies to all *coarse* ferrite microstructures, even if the structure is uniform.

This conclusion is in agreement with A. Le Chatelier's results on brittleness of steel. That eminent authority has shown that in any hypo-eutectoid steel which has been cooled slowly after being heated above Ac_3 , though under slow tensile testing very considerable toughness is exhibited, yet as the speed of testing is increased, the toughness decreases, and when the speed of testing is so great that the ultimate strength of the steel is reached almost instantaneously (shock test), the test bar breaks practically without elongation or contraction of area. On the other hand, quenched and reheated steels, provided the reheating temperature does not exceed 700° C., exhibit as great toughness when tested by shock tensile test as when tested by the usual slow tensile test.

In the table below are given the physical properties and heat treatments of test bars cut from a single casting of steel contain-

¹ Congrès International des Méthodes d'Essai des Materiaux de Construction, Paris, 1900, vol. ii, Part I, pp. 13-25—see especially p. 16, paragraph 4.

ing .10 per cent. carbon, .19 per cent. silicon, and .23 per cent. manganese.

Blowholes made it impossible to secure tensile tests on two of the bars, and one bending bar that undoubtedly should have bent 180 degrees was spoiled by a blowhole. It is particularly to be noted in these tests that the superiority of the heat-treated over the raw steel. as judged by the tensile test alone, is not very great; as judged by the bend, is considerable, without revealing any variation in the toughness of the bars heat treated by the several methods; but as judged by the shock test, the toughness is improved some four-fold in the annealed and slowly cooled bar, seven-fold in the air-cooled and reheated bar, eight-fold in the bar cooled at an accelerated rate, and ten- to eleven-fold in the quenched bars, whether reheated or not. The shock test of 2.5 kgm., given by the raw steel, is extremely low; and the breaking of this specimen without measurable deflection. with a coarse crystalline fracture, confirmed the reading of the machine. The two tests that gave 25 and 27.5 kgm, did not break. It would be difficult to find a more striking example than this of the inadequacy of slow tensile testing alone, or even in combination with a bend test, to reveal even very considerable brittleness. steel of .10 per cent. carbon requires annealing at all, is not admitted by many foundrymen. Yet, as the tests show, not only does it require annealing, if used for any purpose demanding toughness, but if the maximum resistance to suddenly applied stresses is to be secured, castings of even this very soft steel should be quenched. and of course, reheated to relieve stresses (the actual hardening of such a steel by quenching is too slight to affect the usefulness of the castings).

| No. | Treatment Temperature, deg. C. and time | Tensile strength | Elastic limit | Ext. per cent. in 2 in. | Cont. per cent. | Fracture | Bend in. X I in., on I-in. man- drel, deg. | PTP- |
|--------------|---|---------------------|------------------|----------------------------------|-----------------------|---------------------|---|------|
| A | none | 50,020 | 23,930 | 35.2 | 47.85 | 1 | 50 | 2.5 |
| В | 900-3 cool in 40 minutes | 55,390 | 28,750 | 38.5 | 67.0 | cryst. Silky cup | 180 | 20.0 |
| С | 900-3 cool slowly | | | | | | 45 (blow hole) | 10.5 |
| D | 900-3 quench | | | [| | | 180 | 27.5 |
| \mathbf{E} | 900-3 air cool \ | 56,450 | 32,950 | 39.25 | 65.2 | Silky cup | 180 | 17.5 |
| | 710-6 air cool ∫ | | | <i>-</i> | | | | |
| F | 900-3 quench 680-8 air cool | 53,200 | 27,100 | 34.95 | 60.4 | Silky cup | 180 | 25.0 |

Medium carbon steels, especially those containing a large proportion of manganese, are made tremendously strong, and at the same

time very highly resistant to shock, by rapid cooling and reheating. Examples could be given of the results of 12 years' experience with such steels, but only a few will be quoted. They are, however, typical.

Figure 53 shows the microstructure of a steel containing .21 per cent. carbon and 1.17 per cent. manganese, heated to 900°C. for 5 hours and quenched in water, then reheated to 650 to 680°C. for $6\frac{1}{2}$ hours and cooled in air. This microphotograph was taken from a test coupon attached to a casting weighing about 1000 pounds; the physical properties of the bars cut from this coupon were:

| TS | EL | Ext. per cent. in 2 in. | Cont., per cent. | Fracture | Bend | Fremont, kgm. |
|--------|--------|-------------------------|---------------------|----------|------|---------------|
| 91,900 | 71,650 | 20.37 | 50.25 | Silky | 180 | 21.0 |

The very high shock test and bend, coupled with great strength, is particularly noticeable.

The author once had occasion to test two small converter castings; they were too small to cut tensile or bending tests from them, but a bend of a projecting lug of one casting was made, the lug being bent flat on itself without fracture. The analyses and Fremont tests of these castings were:

| No. | С | Si | Mn | S | P | Fremont, kgm. |
|-----|------|-----|------|------|-------|---------------|
| 33 | .18 | .41 | . 93 | .065 | .052 | 5.0 |
| 34 | . 20 | .45 | 1.02 | .057 | . 049 | 7.0 |

The very low shock test value, despite good slow bending toughness, is especially to be noted. The shock tests broke with almost no deflection, and showed a rather coarse crystalline fracture, practically identical in both. This coarse fracture of the shock tests of such steels is typical. The annealing of these castings had been by heating and slow cooling, and was very insufficient in No. 34, as the microstructure showed. Yet, the Fremont tests were about alike, even slightly better for No. 34 than for No. 33.

Casting No. 34 was sawed into two approximately equal parts, and one-half heat treated as follows: Heated to 900°C. for 4 hours and quenched in water, reheated to 680°C. for 8 hours and cooled in air. The two halves, one in the annealed, the other in the heat-treated condition, were then subjected to the blows of a drop of about 500 lb. falling weight, with the result that the annealed half broke in three pieces at the first blow of the drop from a height of

4 ft., showing a coarse crystalline fracture; while the heat-treated half endured without fracture one blow from 4 ft., one blow from 5 ft., one blow from 6 ft., one blow from 7 ft. and two blows from 8 ft.; six blows in all. The two halves of the casting, after test, are shown in Fig. 54, annealed half (three pieces) to the left, heat-treated half (one piece) to the right. In Fig. 55 a back view of the heat-treated half, after test, is shown.

This drop test confirms the result of the Fremont test, which showed that the castings as annealed were brittle under suddenly applied heavy load, though they gave a good bending test. No section of these castings was over 1 in. thick, and the average thickness was about $\frac{1}{2}$ in. The impact test is excellent for revealing brittleness in thin castings of this sort.

A rough test used by the author to demonstrate the toughness of heat-treated cast steel in heavier sections consisted in striking a $2\frac{1}{2}$ -in. square bar of the steel, resting on supports 24 in. apart, with a drop of about 500 lb. weight, falling some 10 or 12 ft. A 2-in. round bar was set in the middle of the test bar to localize the shock, and the test bar reversed after each blow. The result was a sort of accelerated endurance test under shock, the bars being deflected about \frac{1}{2} in. at each alternate blow, and straightened by each alternate blow. Of course, the greater toughness of the outside fibers, which are subjected to the maximum stress, increases the resistance of the bars to fracture in this test. Ouenched and annealed bars of steel of about .25 per cent. carbon and 1.20 per cent. manganese, tested in this manner, have endured from 28 to 57 blows before rupture. What was particularly noteworthy in the performance of these bars was that frequently they exhibited cracks completely across each face, and extending some $\frac{1}{4}$ in, into the bar, three or four blows before final rupture took place, indicating very considerable toughness and resistance to the spreading of the cracks in the interior portions of the test bars.

Another rough test, used to exhibit the toughness of castings treated by quenching and reheating, was adopted for castings having an eye some 6 in. in internal diameter and 2 ft. long, with walls about $2\frac{1}{2}$ in. thick, used to fasten these castings by means of a heavy pin, to another part of the machine. A hard steel wedge of suitable taper was seated in this eye, and then driven down by the blows of a heavy drop, weighing about 1500 lb. and falling about 12 ft. Quenched and annealed steel castings (.25 per cent. carbon, 1.20 per cent. manganese, of about 85,000 lb. tensile strength), so tested,

endured from 8 to 14 blows before rupture; and in some cases the wedge could be driven no further with the drop employed. The rupture of the walls of the eye almost invariably followed an irregular line, the steel tearing slowly apart. No comparison with steel of the same carbon, annealed by heating and slow cooling, was made, but compared with higher carbon castings of approximately equal strength, the results of the tests were most illuminating, the harder steels never enduring over two blows. The second blow on the harder steel castings sent the fragments, or sometimes the whole eye, hurtling across the shop.

That these castings possessed great strength and toughness is demonstrated by the fact that in service they wore down on one side of this eye, which endured very heavy stresses applied somewhat suddenly, from the original thickness of some $2\frac{1}{2}$ in. to a thickness of $\frac{1}{4}$ in., without rupture of the eyes.

The following figures are given to show the physical properties of a number of test bars, selected at random, cut from these castings. Each group represents a lot of tests made on the same day, though not in every case from the same annealing furnace.

| Tensile strength | Elastic limit | Ext. per cent. in 2 in. | Cont., per cent. | Fracture | Fremont, kgm. | Bend ½ in. by 1 in., on 1- in. man- drel, deg. |
|---------------------|------------------|-------------------------|---------------------|-----------|------------------|--|
| 88,850 | 63,500 | 26.60 | 52.9 | Silky cup | 19.0 | 180 |
| 88,300 | 61,240 | 28.11 | 54 - 75 | Silky cup | 14.0 | 180 |
| 89,600 | 60,400 | 26.19 | 47.3 | Silky cup | 15.0 | 180 |
| 85,950 | 60,400 | 28.36 | 59.05 | Silky cup | 22.0 | 180 |
| 89,200 | 61,020 | 26.19 | 51.0 | Silky cup | 11.0 | 180 |
| 100,000 | 70,820 | 23.8 | 49.65 | Silky cup | 23.5 | 180 |
| 92,850 | 62,300 | 24.66 | 55.6 | Silky cup | 15.0 | 180 |
| 90,100 | 60,200 | 26.95 | 58.75 | Silky cup | 13.0 | 180 |
| 98,800 | 66,680 | 22.80 | 48.75 | Silky cup | 24.0 | 145 |
| 88,550 | 60,600 | 26.60 | 54 · 4 | Silky cup | 11.0 | 180 |
| 96,150 | 65,070 | 25.57 | 57.15 | Silky cup | 15.0 | 180 |
| 90,000 | 55,100 | 31.05 | 56.3 | Silky cup | 15.0 | 180 |
| 85,100 | 55,970 | 29.14 | 58.8 | Silky cup | 30. 0 | 180 |
| 80,8 0 0 | 51,510 | 29.14 | 62.5 | Silky cup | 28.5 | 180 |
| 90,700 | 62,780 | 27.20 | 56.6 | Silky cup | 25.5 | 180 |
| 85 , 950 | 51,040 | 25.26 | 58.6 | Silky cup | 30.0 | 180 |
| 83,450 | 51,700 | 29.69 | 60.45 | Silky cup | 27.5 | 180 |

During the war, the author used steel of this composition, heat treated, in the manufacture of castings for the army and navy. An amusing instance of the tremendous toughness of the castings was our experience in trying to break up for scrap a number of pieces which had been condemned on account of a shrinkage cavity, the casting being an extremely difficult one to make perfectly sound. When the castings reached the scrap breaker, trouble arose because they could not be broken, but bent back and forth several times, in a manner most exasperating to the crew of the "skull cracker," whose business was to break a respectable amount of scrap per day. We at once called the attention of the authorities who had rejected the castings to the "test" they were getting at the drop, which made quite an impression, as these castings were of high tensile strength steel, which as furnished by the average foundry would break readily.

In a paper recently published by Dr. F. C. Langenberg, that well-known authority gives a number of tests made upon steels of this special composition, which show that the shock toughness of these steels is five or six times as great as that of plain cast steel of normal manganese content, with a carbon content that gives equivalent strength.

Since the author first described the use of this special composition and heat treatment of cast steel before the American Society for Testing Materials in 1913,² it has been adopted by several foundries for work where extremely tough, strong steel is an absolute essential. In particular one large company has manufactured car coupler knuckles of a steel of this composition, made in the electric furnace, heat treated by quenching and drawing at such a temperature as to give very high strength and great toughness. During the war, this company applied the same steel with great success to cast anchor chain,³ solving the problem of moulding and pouring the links in a way that commanded the un-bounded admiration of foundrymen.

Credit for the original discovery of the beneficial effects of a manganese content over 1 per cent. upon low and medium carbon steels is due to the Bethlehem Steel Company, and especially to the late Maunsel White. That company has manufactured such a steel

^{1&}quot;Impact Tests on Cast Steel," Proc. A. S. T. M., vol. xxi (1921), p. 840.

2 J. H. Hall, "Heat Treatment of Hypo-eutectoid Carbon-steel Castings," Proc. A. S. T. M., 1913, vol. xiii, p. 514.

³ See H. J. Cox, "Cast Steel Anchor Chain," Proc. A. S. T. M., 1918, Part II, vol. xviii, p. 100.

for years, using it for rolled and forged material. The author believes he was himself the first to apply this composition to cast steel.

Endurance Test.—Under ordinary conditions of service, of course, a casting will never be strained beyond the elastic limit, and therefore a casting of soft steel heat treated to give great shock toughness, will not be called upon to exhibit the plastic deformation which it is capable of enduring. A casting used, for instance, for the front axle of a truck or tractor, will have a very long life under ordinary conditions of service, if it is capable of enduring a great number of stresses repeated constantly, none of which exceeds the elastic limit. We know that the higher the elastic limit of a steel, the higher in general will be its resistance to repeated stresses of a given intensity. As the elastic limit rises in proportion to increase in carbon content, it is in general better to use a medium high-carbon steel for an axle than to use a soft steel, under ordinary conditions of service. Experience with locomotive axles has proved this to the satisfaction of engineers.

In certain classes of service, however, allowance must be made not only for ordinary, but for extraordinary, conditions. Thus, for instance, some manufacturers of taxicabs who used a cast steel front axle, made one portion of the casting sufficiently light to either bend or break, and thus save the rest of the car from injury, when it collided with some heavy object. In practice, they found that some of their axles were so bent, straightened and returned to service, several times. Now, for such service as this, in which it is expected that the steel will occasionally be strained far beyond its elastic limit, a steel of great shock toughness and which will endure plastic deformation will be of value in two ways: First, in such an accident, it will be far less apt to break than a hard steel, and hence will in many cases prevent a serious smash-up. Second, such a steel can be bent and straightened more times than a more brittle steel, so that its use will result in a saving of money spent on replacement.

In order that such a steel should possess a good resistance to repeated stresses which do not exceed the elastic limit, it should be heat treated in such a way as to give it the maximum strength. It is true that, by so treating a hard steel, it will be given much greater resistance to repeated stresses of a given intensity than can be given to the soft steel. But the hard steel can never possess great shock toughness.

In the following tables, unfortunately based on but few examples,

it is shown that a soft cast steel, quenched and annealed, will possess a tensile strength and elastic limit equal to those of a much harder steel annealed in the usual manner by heating and slow cooling, or heat treated by air cooling and reheating; and that the endurance of the soft steel so treated, to alternate stresses, at least of this type, is greater than that of the harder steel. Five bars of the harder steel endured an average of 3,805,000 revolutions at a fiber stress of 28,270 lb. to the square inch; and five bars endured an average of 10,451,240 revolutions at 28,270 lb. to the square inch, and 590,560 revolutions additional at 38,870 lb. to the square inch. The four bars of the softer steel endured an average of 10,373,050 revolutions at 28,270 lb. to the square inch, and 823,950 revolutions additional at 38,870 lb. to the square inch. In every case but one, the soft steel gave a better test than the hard steel. These tests were made on the White-Souther endurance testing machine.

| | | | | | | | | Freatment | |
|-----|-----------|------|-----|-------|------|------|-------------------------|------------------|--------|
| No. | Cut from | С | Si | Mn | S | P | Heated to degrees C. | For | Cooled |
| A | 4 in. sq. | .42 | .46 | .73 | | | 900 | 5 hours | аіг |
| | | | | Į. | | | 700 | 5½ hours | slowly |
| В | 4 in. sq. | .44 | .38 | .59 | | | 900 | 5 hours | air |
| | | | 1 | 1 | 1 | | 690 | 3 hours | slowly |
| C | 4 in. sq. | . 36 | .54 | .71 | .049 | .051 | 920 | 5 hours | air |
| | | | | | | | 690 | 3 hours | slowly |
| D | 4 in. sq. | .47 | .56 | . 7.3 | | | 900 | 5½ hours | аіг |
| | | | | 1 |) | | 700 | 11 hours | slowly |
| E | 4 in. sq. | .43 | .46 | .65 | | | 900 | 4 hours | air |
| | | | | Ì | | | 700 | 5 hours | slowly |
| F | 11×21 in. | . 23 | .31 | 1.07 | .037 | .047 | 900 | 6 hours | water |
| | | i | ł | ļ | İ | | 6 8o | 8 hours | air |
| G | 4 in. sq. | . 27 | .46 | .71 | | | 900 | 5 hours | water |
| | | | | | | | 68o | 81 hours | аіг |

| No. | T S | EL | Ext. in 2 in., per cent. | Cont., per cent. | Fremont |
|-----|--------|--------|-----------------------------|------------------|---------|
| | 82,100 | 45,990 | 8.41 | 9 · 79 | 7 · 5 |
| В | 76,170 | 42,990 | 14.64 | 15.11 | 5.0 |
| С | 73,200 | 40,630 | 14.08 | 19.79 | 10.0 |
| D | 79,590 | 44,020 | 10.94 | 11.49 | 8.5 |
| E | 73,200 | 40,200 | 12.5 | 16.4 | 7 · 5 |
| F | 67,200 | 44,400 | 14.19 | 31.3 | 32.0 |
| G | 78,710 | 46,400 | 22.8 | 24.21 | 17.0 |

| ENDURANCE T | 4 | ĸ. | Α | ľ | N |) (| | k | | ľ | 100 | VI. | ٠ |
|-------------|---|----|---|---|---|-----|--|---|--|---|-----|-----|---|
|-------------|---|----|---|---|---|-----|--|---|--|---|-----|-----|---|

| No. | Fiber stress | Deflection | Revolutions No. 1 end | Revolutions No. 2 end |
|-----|--------------|------------|-----------------------|-----------------------|
| A | 28,270 | . 06 | 10,709,600 | 10,709,600 |
| В | 28,270 | .06 | 4,685,000 (B) | 6,708,000 (B) |
| C | 28,270 | . 06 | 3,474,300 (B) | 2,368,700 (B) |
| D | 28,270 | . 06 | 10,158,400 | 1,789,400 (B) |
| Е | 28,270 | .06 | 10,339,300 | 10,339,300 |
| F | 28,270 | . 06 | 10,475,100 | 10,475,100 |
| G | 28,270 | . 06 | 10,271,000 | 10,271,000 |

| No. | Fiber stress | Revolutions No. 1 end | Revolutions No. 2 end |
|--------------|--------------|-----------------------|-----------------------|
| A | 38,870 | 280,800 (B) | 302,000 (B) |
| В | | | |
| C | | | |
| \mathbf{D} | 38,870 | 1,828,600 (B) | |
| \mathbf{E} | 38,870 | 366,000 (B) | 175,400 (B) |
| F | 38,870 | 393,900 (B) | 845,500 (B) |
| G | 38,870 | 834,100 (B) | 1,222,300 (B) |

B signifies bar broke.

As already stated, it would no doubt have been possible by quenching and annealing the hard steel, to make it superior to the soft steel in this test, but the heat treatment would not have given the hard steel a much greater resistance in the impact test. Hence, although an axle made of the hard steel quenched and reheated, would probably give longer life under ordinary conditions of service than the soft steel similarly heat treated; yet in case of a smash-up, there is little doubt that the hard steel would be broken short off by a blow which the softer steel would endure without rupture. order, therefore, to have in our machinery parts a steel which will resist both repeated light stresses and occasional heavy stresses, we should use a comparatively soft steel which has been quenched and annealed. The front axle is of course a very special case, and is seldom made of cast steel. It is referred to here, however, as a striking example of a type of service for which many machinery castings are made.

Fatigue tests on steel of about .25 per cent. carbon, 1.25 per cent. manganese, made by the author during the last few years, indicate

that the life is very considerably greater in the quenched and annealed condition than when the steel is air cooled and reheated, or given a plain annealing.

Experience with marine engine shafting and steam hammer piston rods has shown in many cases that a heat treatment which gives the steel great shock toughness will increase the life of the forging even as much as five- or six-fold.¹ What is true of forgings will be equally true of castings, except that because a casting almost inevitably contains small blowholes or flaws, it can never be made as reliable as a forging, so that the lengths of life of a number of similar castings will not be so nearly equal as would be true of forgings. But to maintain, as some quite competent engineers do, that because a casting is a casting, and almost sure to contain flaws or blowholes, it is not worth while to heat treat cast steel to give it great toughness, is most illogical. It is certainly poor reasoning to say that because a casting is almost sure to contain a starting place for a crack, it is not worth while to treat the steel to make it as resistant as possible to the starting and spreading of that crack.

High-carbon and Alloy Cast Steels.—Hyper-eutectoid cast steels, that is steels containing over .89 per cent. of carbon, and therefore consisting of pearlite and free cementite, are very little used. Their heat treatment has, therefore, not been worked out with the same thoroughness as that of mild steels. It follows from a consideration of the carbon-iron diagram that the heat treatment of these steels must be similar to that of hypo-eutectoid steels. Annealing must be at a temperature above the line SE in order to dissolve the cementite in the austenite, and the effect of varying the rate of cooling should be quite similar. The researches of Boynton² on hyper-eutectoid rolled steels have shown that by hastening the cooling from above the line SE, the separation of cementite can be largely suppressed. The temperatures for reheating should be the same as for hypo-eutectoid cast steels.

Chrome steels containing about 1 per cent. chrome, nickel steels of $2\frac{1}{2}$ and $3\frac{1}{2}$ per cent. nickel, and nickel-chrome cast steels can be annealed and heat treated in the same manner as can carbon steels, using the temperatures that are suitable for ordinary carbon steels. Though the chrome and nickel-chrome steels do not coarsen on slow cooling from above Ac_3 as markedly as do the carbon steels, yet the

¹ Prof. Henry M. Howe, discussion of the author's paper "Shock Tests of Cast Steels" referred to above.

² Iron and Steel Magazine, May, 1904.

beneficial effect of heat treatment by air or water cooling and reheating upon these steels is almost if not quite as great as in the case of the carbon steels, and whenever the toughest and strongest possible steel is desired, heat treatment should be resorted to. Owing to their coarsening on air cooling less than do the carbon steels, nickel steels air cooled and annealed can often be given properties as good as those of quenched and reheated carbon steels. Castings too large to be quenched with safety, therefore, may be made of nickel steel and heat treated by air cooling and reheating with excellent results. In Giollitti's recent book¹ are shown many examples of this practice.

The addition of nickel to steel lowers the critical points to a degree proportional to the amount of nickel added. The annealing and reheating temperatures, therefore, can be lowered to a corresponding degree, though it is by no means essential to do so. But as the sluggishness of the molecular changes in steel increases as temperature falls, the time of exposure to these temperatures should be increased in proportion as the temperatures themselves are lowered.

Manganese steel, containing, as usually made, from 10 to 15 per cent. manganese, and from .80 to 1.5 per cent. carbon, is heat treated by methods quite different from those followed for ordinary cast steel. By the combined effects of carbon, manganese and rapid cooling, manganese steel is rendered entirely austenitic and very tough, because the transformations are nearly or quite suppressed. In the cast condition, or when heated and slowly cooled, this steel is very brittle, owing to partial transformation of the austenite, and to the liberation of free carbide of iron and manganese (manganiferous cementite?), between the austenite grains.

As the steel is quenched in water from a high temperature, generally over 1000°C., the castings, if of unequal section, are left after treatment in a state of heavy stress. Relief of these quenching stresses is, moreover, extremely difficult, as to reheat the steel to a temperature high enough to relieve them, results in great brittleness of the steel.

The brittleness of the steel when cast; its high shrinkage, which results in heavy stresses in cooling in the mould and in treatment; and its low thermal conductivity; these properties make the design and heat treatment of manganese steel castings of complicated shape a difficult and fascinating task.

1"Heat treatment of Soft and Medium Steel"—F. Giolitti, translated by Thum and Vernaci.

Annealing Furnaces.—Annealing furnaces are built in a great variety of designs, and almost any fuel may be burned in them. Coal, fuel oil, natural gas, or even occasionally producer gas, may be used, depending upon the price of the fuel. Producer gas, however, which requires regeneration for its combustion, is not commonly a good fuel for this purpose, since annealing furnaces generally have to be cooled off between heats, and often lie idle for many hours. Such conditions interfere seriously with the operation of a furnace that is at its best only when run continuously, and prevent the attainment of the economy that regenerative furnaces will give under suitable conditions. In spite of the high cost of electric heating, the electrically heated annealing furnace is being widely introduced, and a number have been installed in steel foundries. Their advantages are accurate control of temperature, comparative freedom from scale on the castings, due to the absence of oxidizing gases in the furnace, and increased life of the refractories. chief disadvantage is that electric heating generally is more expensive then fuel; in some cases this can be partially offset by the use of a continuous furnace, in which the castings are introduced at the cold end of the furnace and gradually moved up to the hot end.

In general, annealing furnaces should not be built large enough to take the entire day's output of a foundry at one charge, particularly if the size of the castings varies considerably. The practice of charging large and small castings in the same furnace is a very poor one, since if time enough is given the large castings to heat through properly, the small ones suffer from oxidation and consequent scaling; and a real loss is experienced in that the small castings are heated much longer than necessary, at a considerable expense in fuel.

Very large annealing furnaces, heated by one or more fire boxes so small that the temperature in the furnace will vary considerably from point to point, are most inefficient for two reasons, if not more. First, the castings at the "cold" points are seldom properly annealed; second, those at the "hot" points are annealed longer than need be, in an effort to bring the "cold" castings up to a proper heat.

Considered from this standpoint, the large "catch all" annealing furnace is most uneconomical, and it is extremely doubtful if the saving in labor attained by its use is sufficient to offset its great wastefulness in fuel and the poor quality of its work. One man can fire three small furnaces nearly as easily as one large one—quite as easily if oil or gas fuel is used—and if the three or more furnaces are designed with a view to making them suitable to the size of castings

that are to be annealed in them, an economy of fuel will result from their use.

The deep furnace with a lift-off top gives great economy in charging labor, and when used for castings that are to be annealed by slow cooling, or accelerated cooling, does fairly good work. If heat treatment, properly so-called, is to be used, so that the castings have to be taken from the furnace for quenching or air cooling, it will frequently be necessary in such a furnace to set them on a heavy steel plate that can be lifted out. To remove castings one at a time from an open-topped furnace will not be very easy; and if they must not be allowed to cool before quenching, it will be hard to get them all out in time. Though the castings are generally piled in these furnaces more or less haphazard, it will in most cases be best to set them up more carefully so that they can be heated evenly and will not suffer from warpage.

The furnace with a movable (car) bottom is in most cases the best type to install, especially when heat treatment is to be used. They are more costly to build than open-top furnaces, but can be charged quite as cheaply, and make it easy to withdraw the castings one at a time for quenching, without cooling off the rest of the charge; or to take out a number of comparatively small castings set on plates and quench them in batches. A furnace with fixed bottom and a door at the front can of course be used for the same work, by charging and drawing with piels, but the labor expense in such work is very great.

Continuous furnaces may be used for the heat treatment of very light work, if there is a great deal of it to be done. Such furnaces are often used to anneal small forged parts, and can of course be utilized to equal advantage for small castings.

Pyrometers.—Temperature control in steel casting annealing has long been one of the weakest points in the practice of the shops, and lack of proper precautions to ensure correct annealing temperature has been almost universal. So common has been the practice of annealing steel castings without control of temperature, that in the majority of shops heat treatment has had no chance of success.

The great majority of carbon steel castings require only annealing by heating and slow cooling; or at most, by accelerated cooling, followed by slow cooling to relieve strains. For such work, it may sometimes be possible to estimate the temperature closely enough by the eye, but when the eye is relied upon it will be found that no two heats will be annealed quite alike, because the man is not yet born who can recognize small temperature intervals by color. For the heat treatment of ordinary steel, the heating of alloy steels, and all work where the temperature must be closely controlled, pyrometers are absolutely indispensable; and their general use is advisable.

In practice, however, the use of pyrometers is not an absolute guide to the temperature of the castings, since it is neither economical nor sensible to install enough thermocouples in each furnace to give its temperature throughout. The eye must always be relied upon to a great extent to make sure that the steel throughout the furnace has reached the correct temperature. With one or two thermocouples in the furnace, the fireman can heat his steel quite uniformly to a prescribed temperature, because he can *match* colors very well indeed, and can bring all the castings to the color of the spots where the couples are. In this way he can do far more accurate work than is possible even by the use of a painted color scale; such a scale is not as useful as might appear, because the color sensation produced on the eye by a hot object is not capable of being matched closely against that produced by a painted board.

To do really good annealing, it is essential that the fireman be able to see into his furnace by means of properly arranged peep holes, so that he can see not only the castings, but the thermocouples. In this way he can be sure that the castings are heated to the temperature indicated by the pyrometer. Whether a pyrometer is used or not, such peep holes should be provided to enable the operator to see that his castings are evenly heated. Far too often, we see furnaces in which pyrometers are installed, but peep holes are not, so that the fireman has to rely upon the pyrometer entirely, with no possible way of knowing if the furnace is at an even heat throughout. Had the author to choose between two furnaces, one provided only with a pyrometer and one only with peep holes, he would unhesitatingly choose the latter, but that both should be provided would seem to require no argument.

Recording pyrometers are a great aid to the metallurgist, as they give him an absolute record of the firing of any furnace. They may be located in the superintendent's office, where the men do not have access to them, in which case a single instrument that can be attached to any one of a number of furnaces may be used. In such a system, the instrument is relied upon as a sort of policeman to make the men pay attention to their firing and heat to such temperatures as are ordered.

A far better way of using recorders, however, is to locate them in

dust proof cases at the furnaces, preferably with an instrument for each furnace or a multiple-point recorder to take care of a battery of furnaces. When the temperatures are marked up on a sheet for the men to see and watch, they take far more interest in doing good work, as an irregular curve for their fellows to laugh at serves as a spur to keep them up to the mark. Having tried both the private detective method and the "cards on the table" method, as it were, the author strongly recommends the latter.

The men, however, should not be allowed to fall into the habit of simply heating up the furnace until the pyrometer indicates the correct temperature, holding the pyrometer at that temperature the proper length of time, and then cooling off the furnace or drawing the castings, as the case may be. Once taught to rely entirely upon the pyrometer, they consider that the burden of responsibility for the proper annealing of the steel has been shifted from their shoulders to those of the man who standardizes the pyrometers, and they no longer take pains that the steel shall all be at the temperature desired. Perhaps the best way to secure their attention to their work is by the use of test bars attached to various castings, or charged with the steel at several points in the furnace. A few test in every heat have a wonderful effect in making the men "sit up and take notice," and even if they are never tested at all (so long as the men do not find it out), are extremely useful. Used for tests, recorded in such a way as to be readily consulted and compared, they are invaluable.

No pyrometer system yet invented is "fool proof," nor a substitute for care and conscientious work on the part of the fireman, nor an absolute guide to correct annealing and heat treatment, but properly handled, such an installation is so useful and removes so much uncertainty that after a few months' use it appears well-nigh indispensable.

CHAPTER X

FINISHING, STRAIGHTENING AND WELDING

Castings frequently become warped to a certain extent, either from shrinkage in solidifying, or from bending when hot in the annealing furnaces, and then have to be straightened. As far as possible, this should be taken care of in the design of the pattern. instance, if a projecting lug which should be at right angles with the casting is found to consistently take a position at an improper angle, it is far simpler to shift the lug on the pattern in the opposite direction, so that the lug on the finished casting shall be at the correct angle, rather than to bend the casting. Frequently, however, the bending of steel castings cannot be avoided, and the best way to do the work has to be considered. If the steel is sufficiently ductile, the part to be bent not too thick, and the degree of bend not too great, bending cold can be practised without harm to the casting. In too many shops this cold straightening is done with a drop of some kind, a practice that cannot be too heartily condemned. effect of the suddenly applied blow is necessarily largely local, instead of being spread out along the casting, and frequently results in the cracking of the work. The crack may be visible, and spoil the piece; or even worse, it may be concealed and appear later in a broken casting whose failure results in loss of time, money, and even life. Even very tough alloy steels are frequently injured by this practice, and many failures of alloy steel castings have been traced to abuse in straightening. The hydraulic or power press, which enables the bending to be carried out slowly, is the proper instrument for bending castings and should be used to the absolute exclusion of bending by blows, except in the case of wide thin pieces which sometimes cannot be satisfactorily straightened under a press.

Bends in hard and comparatively brittle steels, or heavy bends, especially of thick sections, can be made only while the castings are hot. Experiments have shown that the best temperature for doing this work is from 300° to 600°C.—below 300° ordinary cast steel is not sufficiently softened by heating to bend easily, and above a dark-red heat the castings are naturally difficult to handle. If much work

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has to be done, however, operations may have to be begun at about 600°C. in order to complete the task before the casting grows too cold to bend readily. Manganese steel, however, should not be heated above 300° to 350°C., once it has been treated, as temperatures above this make the steel brittle.

Should the casting have been heat treated by rapid cooling and reheating, or by accelerated cooling, it is desirable not to heat it again above 720°C., since temperatures higher than. 720° destroy the heat treatment, producing coarse ferrite and reducing the ductility of the steel. Steel annealed in the ordinary manner by slow cooling, may if necessary be heated for straightening to any temperature below the annealing heat, without damaging the heat treatment. In case these higher temperatures are used for straightening, however, it is desirable to allow the castings to cool slowly after straightening, or heavy shrinkage stresses will be set up in the metal, neutralizing to a certain extent the benefit of the annealing.

In particular, the practice of heating only one part of a casting in a bed of coals or a small furnace, straightening it, and allowing it to cool freely in the air, is to be avoided as far as possible, because the stresses set up in the piece by the local heating and rapid cooling may be as heavy as those existing in the casting before it was annealed, and may well result in trouble when the casting is put to work. If a coal fire or small furnace must be used for local heating of castings, the least that can be done is to allow the steel to cool slowly at one side of the fire, or in the furnace, after it has been straightened. In the majority of cases, however, heating the whole casting is the best practice, and as far as possible should be resorted to.

Welding.—Blowholes, shrink holes (as for instance under sink heads), etc., which not many years ago would have made it necessary to scrap a casting, are to-day welded up by means of the oxyacety-lene or other gas torch, or the electric arc. In filling up these holes, it is absolutely essential to heat up the entire surface of the hole before any metal is run in. To do this, it is frequently necessary to enlarge the top of the hole to a considerable degree, in order to get at the lower portions. Too frequently, the hole is merely run full of metal which does not weld to the casting except at the surface of the work, resulting in a job little better than can be done by pounding cold metal into the hole—a mere device to cover up bad work and get it past the customer's inspectors.

In the hands of a skilfull welder, "shrinks," sand holes, small scabs,

etc., are repaired in such a manner as not only to remedy the defect, but to produce a casting nearly, if not quite, as serviceable as though it had been sound in the first place. Small areas of surface blowholes or blowholes that appear on machining, may be similarly patched up; to do a really satisfactory job, the affected part should be cut out completely and new metal welded in to replace it. If the blowholes are found to cover more than a small proportion of the casting, however, welding should not be attempted; it is cheaper in the long run to scrap the piece.

Cracks and tears in castings, if not too large or so located as to weaken the piece to too great an extent, may be successfully repaired by welding, though too much credence should not be given to the claim of the welder that he can make the casting as strong as a sound one. True, the art of autogenous welding has been developed to a very great extent in the past 10 years, and one even sees published tests purporting to show that the weld is as strong as, if not stronger than, the unwelded metal. The truth, however, is that even a very good autogenous weld cannot be expected to have over 80 per cent. of the strength of the unwelded steel, and that many welds do not approach that figure. Then, too welders vary greatly in skill, and even the best of them turn out a certain proportion of defective work, which can seldom be detected by surface inspection. Very good judgment, therefore, is needed in determining whether to repair a cracked or torn piece by welding, or to make it over.

Either the electric arc or the gas-torch may be employed for welding small cracks and tears, the latter being much more generally used. For very heavy jobs the thermit process is the best to employ. In this method of welding, the defective metal is entirely cut away, a sand mould is built up around the place to be welded, with runner, gate and head as in regular moulding practice, and the very hot thermit steel is then run into the mould, fusing the metal with which it comes in contact and uniting with it. If the thermit metal is properly compounded it may be given practically the same composition as that of the rest of the casting. The same, of course, may be true of the metal deposited in arc or gas welding, though commonly such welds are made with low-carbon steel.

The welded metal is deposited at so high a temperature, however, and has in consequence so coarse a crystalline structure, that even with very careful annealing its all-around physical properties are not as good as those of good cast steel. No welded casting, therefore,

should be considered as strong as one that was sound in the first place and required no repairing.

In the case of all but the softest steel castings, the steel of the weld itself, and the casting adjacent to the weld, are hardened to a remarkable degree in cooling in the air. This is due probably to the high temperature to which the metal is heated by the flame or the arc, which results in a very rapid cooling through the critical range, the heat being absorbed from the hot portions both by the air and by the cold parts of the casting. Under these circumstances, an air cooling becomes the equivalent of a water quenching, hardening the weld so much that it cannot be machined. For this reason all castings that have to be machined on the welded face, and all castings in which large and important welds have been made, should be annealed after welding. Hence, it is desirable to do as much of the welding as possible before the castings are annealed. Surface defects, therefore, and small cracks and tears are almost always repaired before annealing.

In the case of larger cracks, it is often necessary to anneal the casting before attempting to make a weld, because the stresses that caused the crack are still present in the unannealed casting, and will in many cases tear the weld apart again as the casting cools. In making these large repairs, the piece is commonly preheated for some distance on each side of the weld, in order to prevent the weld tearing apart after it is finished, and in many cases this precaution will take care of the unrelieved stresses in the raw casting; but even this is sometimes insufficient, and annealing before welding is then essential for the complete relief of the stresses set up in cooling in the mould. It is best to anneal the piece again after welding, not only to make the welded part machinable, but to relieve the stresses set up as the weld cools off, and to give the metal in and near the weld the greatest possible toughness.

CHAPTER XI

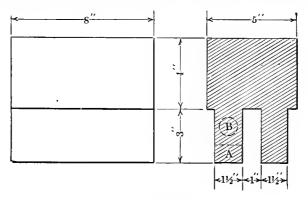
LABORATORIES

To-day, the steel foundryman who should attempt to run his ship without chemical analysis of his raw materials and product, would be in danger of incarceration as a lunatic. Yet it is but a few years since steel makers scoffed at the idea of employing a chemist. The very small foundry, of course, may not have its own chemical laboratory, and may analyze only occasional heats; nevertheless, chemical analysis is an essential in handling the shop.

"An ounce of prevention is worth a pound of cure," is a maxim that the steel foundryman may well keep in mind. In the conduct of the shop, systematic use of the chemical laboratory to keep record of the analysis of raw material used and steel made, from day to day, is worth a hundred "investigations" started after trouble has developed in the foundry, or in the machine shops of the customer. Whatever analysis of the steel is deemed necessary, should be made and recorded every day, and the records kept in such a manner that if trouble occurs there will be some information to go on, in searching for the cause of the difficulty. The first result of maintaining such a system will be that trouble will be easier to account for; the second, that trouble will be less frequent.

As a general rule, the chief chemist should not be the man who does the calculating for the melting shop. In the large shops, of course, this is seldom the case. But small steel foundries frequently have but one man on the premises who is capable of doing this work, and that man the chemist. The human being may some day be born who will be capable of figuring a heat of steel, and gazing undismayed upon analytical results for that heat that show a composition that was not desired. But, allowing for the little weaknesses of human nature, it is best to have one man calculate the steel, and another analyze it.

Just as our fathers doubted the value of chemical analysis in steel making, so this generation questions the usefulness of microscopic examination and physical testing of their product, as a part of the shop routine. Some decry these methods altogether, others say they are of value solely as "post mortems." The author's experience in this line has been, that just as systematic records of analysis are of value not only in locating the causes of trouble, but in preventing it, so daily checking up of the work of the shop, particularly the annealing furnaces, by means of test bars, will keep the product uniformly excellent and lead to many improvements. When a certain amount of information, as for instance tensile test, bending test, microstructure and perhaps shock test and hardness, is obtained each day from test bars in every annealing heat, or a certain number of heats,



A-Bend Test B-Tensile Fig. 56.

it will not be long before enough data are accumulated to show the probable cause of the poor quality of a broken casting. Moreover, once this information is at hand, a poorly annealed heat can be recognized and held up before it gets out of the shop; and by investigation of one's best and poorest product, as well as that of competitors, much improvement will almost certainly result. The metallographist is rapidly winning his place in the steel business, and when the dust clears away, it will be found that his work is parallel with that of the chemist who is now no longer called on simply to conduct autopsies, but has his recognized place in regulating things so that autopsies shall not be needed.

It is the author's experience that test coupons upon ordinary steel castings should be attached to the casting along the full length of one of their long sides, and should be placed in the drag, preferably at the very bottom of the casting. So located, the coupon is properly fed by the casting above it, and will be perfectly sound. In order to secure a clean coupon, it is frequently cast in a special core bedded in the sand of the mould. Such a test piece should be about 3 in. deep, $1\frac{1}{2}$ in. thick and 6 to 8 in. long, attached to the casting by the $1\frac{1}{2}$ - by 8-in. side.

When the casting is too small to feed such a test piece, a proper coupon must be cast either in a separate mould or core or in the same mould as the casting and attached to the latter by a suitable gate. It is generally best to cast these coupons in a dry sand core. The commonly adopted design for these test pieces is shown in Fig. 56.

CHAPTER XII

"BUILDING UP" IMPURITIES IN STEEL

In a number of places in this volume calculations are submitted showing the maximum increase in non-oxidizable impurities of the steel, that is to be expected after making a great number of heats, each of which contains a certain proportion of scrap from previous heats. These calculations are based upon the method worked out by A. H. Jameson, the gist of which is given here for reference. For the full text of this article see *Iron Age*, February 23, 1911, p. 480, and August 21, 1913, p. 406.

"It should be remembered that, through the melting and converting losses, we do condense in the steel any element which does not itself suffer loss in the melting or conversion. But this condensation takes the form of an increase in the percentage by gradually decreasing increments, and theoretically the increase goes on for an infinite number of operations. But it approaches a limit, even though it never reaches this limit, and if we plot a curve to show the increase in percentage by ordinates and the number of operations by abscissæ we find that at infinity the curve becomes a straight line."

"Practically we are interested in the limit only. We wish to know what will be the maximum content in the steel after a long run, and this we can determine readily."

"Let a = the percentage of steel scrap in the cupola charge.

100 - a = the percentage of pig iron in the cupola charge.

b = percentage of loss in both melting and converting.

c =percentage of the element in question in the pig iron used.

x = the maximum percentage of the element in the steel after an infinite number of operations.

"When the limit is reached, and the curve becomes a straight line," we shall find, of course, that after any operation our steel contains exactly the same percentage of the element in question that was in the steel scrap which was charged to the cupola furnace as stock for that blow, so that

$$x = \frac{(100 - a)c + ax}{100 - b}$$

and by resolving

$$x = \frac{(100 - a)c}{100 - (a+b)} \text{ or } \frac{100c - ac}{100 - (a+b)}$$

"It will be noted that the foregoing formula and calculation are applicable to an element which, like copper, gains nothing from outside sources. Now, let us see what is the problem in the case of sulphur or phosphorus, which are increased by the amount absorbed from the fuel in the melting. Let d = the percentage of the element which is taken up by the metal in one melting operation; this can be determined readily by analysis of the stock before and after melting, and is a constant for a given fuel and burden. Then our original formula will become

$$x = \frac{(100 - a)(c + d) + a(x + d)}{100 - b}$$

and by resolving,

$$x = \frac{(100 - a)c + 100 d}{100 - (a + b)} \text{ or } \frac{100(c + d) - ac''}{100 - (a + b)}$$

Mr. Jameson's method is manifestly applicable to any method of melting in which the percentage of a non-oxidizable impurity is increased by melting losses, with or without "pick up" from the fuel: or, for that matter, to a case where by gain of weight in melting (owing for instance to ore additions), the percentage of the element is decreased, with or without a countervailing gain from "pick up." In this volume, the particular method of using these formulæ has been varied from place to place, for greater convenience. for instance, it is permissible to substitute for percentages of pig iron, scrap, etc., and percentages of metal obtained, the actual weights of material charged and of the metal obtained. In certain cases where a "pick up" occurs, the figuring has been done by multiplying the weight of each portion of the charge by the percentage of the element in question in that portion, adding the products, adding to this the weight of the whole heat multiplied by the percentage of "pick up" of the element in question, and dividing the sum total by the weight of the heat. That these variations in the method of using the equations are permissible, is clear at once. Thus the reasoning has been applied to all methods of steel making by the acid process, and to special mixtures of raw materials. influence of the recarburizers, which often contain high percentages of phosphorus, has also been shown in a number of cases; and the method has been used to work backward from a known permissible maximum of impurity, to ascertain the percentage of impurity in the raw material that cannot be exceeded without producing steel above the desired maximum.

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